[redacted]

9

International Correspondence Schools

I. C. S.

REFERENCE LIBRARY

A SERIES OF TEXTBOOKS PREPARED FOR THE STUDENTS OF THE INTERNATIONAL CORRESPONDENCE SCHOOLS AND CONTAINING IN PERMANENT FORM THE INSTRUCTION PAPERS, EXAMINATION QUESTIONS, AND KEYS USED IN THEIR VARIOUS COURSES

GASES MET WITH IN MINES
MINE VENTILATION
HOISTING AND HOISTING APPLIANCES
SURFACE ARRANGEMENTS AT BITUMINOUS MINES
SURFACE ARRANGEMENTS AT ANTHRACITE MINES

16- 190

SCRANTON
INTERNATIONAL TEXTBOOK COMPANY

27

PRINTED IN THE UNITED STATES.

BURR PRINTING HOUSE,
FRANKFORT AND JACOB STREETS,
NEW YORK.

27 86

CONTENTS

INDEX

NOTE.—All items in this index refer first to the section and then to the page of the section. Thus, "Air 5 15" means that air will be found on page 15 of section 5.

D

E

T

U

V

W

GASES MET WITH IN MINES.

CHEMISTRY.

COMPOSITION OF MATTER.

828. Chemistry is that branch of science which treats of the composition of substances and the alterations they undergo in their composition by a change in the kind, number, and relative position of their atoms.

829. Mass and Volume.—The mass of a body is the amount of matter contained in it.

The volume of a body is the space which it occupies. If a body of irregular shape be plunged into a cylindrical jar of water, the rise of the water in the jar, multiplied by the area of its cross-section, will give the exact volume of the body. *Volume is always equal to displacement.*

830. Density is compactness of mass, and has reference to the amount of matter in a given volume of a body. Thus, there is more matter in a cubic foot of iron than in a cubic foot of water; therefore, we say iron is more dense than water; likewise, carbonic acid gas is more dense than air.

831. Specific Gravity.—The specific gravity of any body whatever—solid, liquid, or gas—is the measure of its density. And, in order to measure anything, we must have a standard, or unit, of measure. The standard, or unit, by which we measure the density of all solids and liquids alike is water. In like manner, the unit of measure for all mine gases is air. The chemist uses hydrogen gas as his unit of measure for gases.

Our units of measure, then, are as follows:

For solids and liquids, 62.5 lb. = weight 1 cu. ft. water.

For gases, .0766 lb. = weight 1 cu. ft. air (temperature, 60° F.; barometer, 30").

For example, if we wish to measure the density of iron, we must first know the weight of 1 cubic foot of the iron, and we then find how many times our unit of measure is contained in this weight, which will give us the density (specific gravity) of the iron. Thus, we know the weight of a certain kind of iron is 480 pounds per cubic foot, and we wish to determine its specific gravity. Applying our measure of the density of solids, we find that $\frac{480}{62.5} = 7.68$ is the specific gravity of this iron.

The student must notice carefully that the specific gravity of a body is always the ratio between the weights of equal volumes of the body and of the unit or standard. For this reason, if we take any equal volume of the body and of the unit, or standard, and divide the weight of the one by the weight of the other, we will obtain the same ratio or specific gravity. Now, if we take any irregular piece of coal or other substance, and having first weighed it in the air, we then weigh this same piece of coal in water, the coal will be buoyed up by the weight of the water which it displaces. Hence, the amount the coal loses, when weighed in water, is the same as the weight of its own volume of water. Therefore, it is evident that if we divide the weight of any solid when weighed in air by the loss of weight when weighed in water, we shall obtain the same ratio, which is the specific gravity of the substance.

From the foregoing, we have the following rule to find the specific gravity of any solid:

Rule.—*Divide the weight of the substance in air by the difference between its weight in air and its weight in water; the quotient will be the specific gravity of the substance.*

Let W = weight of the substance in air;

W_1 = weight of the substance in water;

Sp. Gr. = specific gravity of the substance.

Then, $$\text{Sp. Gr.} = \frac{W}{W - W_1} \qquad (1.)$$

EXAMPLE.—The weight of a piece of coal in the air is 7.62 lb.; when weighed in water, it weighs only 1.62 lb. What is its specific gravity?

SOLUTION.— $\frac{7.62}{(7.62 - 1.62)} = \frac{7.62}{6} = 1.27$. Ans.

Or, if we know the weight of 1 cubic foot of a substance—solid, liquid, or gas—and we divide its weight per cubic foot by our unit of measure, the result will be the same as the specific gravity. Thus, we have the following rule for finding the specific gravity of any solid, liquid, or gas when the weight per cubic foot is given.

Rule.—(*a*) **For Solids or Liquids.**—*Divide the weight per cubic foot of the solid or liquid by the unit weight of the standard (weight of 1 cubic foot of water, 62.5 pounds); the quotient will be the specific gravity of the solid or liquid.*

(*b*) **For Gases.**—*Divide the weight per cubic foot of the gas by the unit weight of the standard for gases (weight of 1 cubic foot of air, temperature 60° F., barometer 30 inches); the quotient will be the specific gravity of the gas.*

Let w = weight of 1 cubic foot of the solid, liquid, or gas. Then, we have

(*a*) For solids or liquids,

$$\text{Sp. Gr.} = \frac{w}{62.5}. \qquad (2.)$$

(*b*) For gases,

$$\text{Sp. Gr.} = \frac{w}{.0766}. \qquad (3.)$$

EXAMPLE.—Take the weight of a cubic foot of mercury as 850 pounds (at normal temperature and pressure), and find its specific gravity.

SOLUTION.— $\frac{850}{62.5} = 13.6$. Ans.

EXAMPLE.—If the weight of a cubic foot of carbonic acid gas at a temperature of 60° F. and a pressure of 30″ of mercury is .117129 pound, what is its specific gravity?

SOLUTION.— $\frac{.117129}{.0766} = 1.5291$. Ans.

832. To make plain the true relation of the terms mass, volume, density, and specific gravity, let us suppose we have 1,000 cubic feet of air at the ordinary atmospheric pressure. If this pressure is increased to two atmospheres, the volume will be reduced to one-half of what it was. The mass, however, will not be changed, because the quantity of matter is still the same; but the density of the air is doubled, and the specific gravity is doubled, since each cubic foot of air contains twice the mass that it contained before compression, while the reduced volume of 500 cubic feet of air contains the same mass that was contained in the 1,000 cubic feet.

The practical use to which the specific gravity of a body is applied is to calculate the weight of a given volume of the substance. For example, the weight of a cubic foot of water is 62.5 pounds, and if the specific gravity of a sample of bituminous coal is 1.27, then the weight of a cubic foot of bituminous coal will be $62.5 \times 1.27 = 79.375$ pounds.

Rule.—(*a*) ***For Solids or Liquids.***—*Multiply the weight of one cubic foot of water (62.5) by the specific gravity of the solid or liquid; the product will be the weight of one cubic foot of the solid or liquid.*

(*b*) ***For Gases.***—*Multiply the weight of one cubic foot of air (.0766 pound), temperature 60° F., barometer 30", by the specific gravity of the gas; the product will be the weight of one cubic foot of the gas.*

(*a*) For solids or liquids,

$$w = 62.5 \times \text{Sp. Gr.} \quad \textbf{(4.)}$$

(*b*) For gases,

$$w = .0766 \times \text{Sp. Gr.} \quad \textbf{(5.)}$$

EXAMPLES FOR PRACTICE.

1. What is the weight of a cubic foot of anthracite coal having a specific gravity of 1.55? Ans. 96.875 pounds.

2. Find the weight of 100 cubic yards of earth having a specific gravity of 1.75. Ans. 147.656 tons.

3. What is the weight of 200 cubic feet of carbonic acid gas at a

temperature of 60° F. and a barometer pressure of 30 inches, the specific gravity of the gas being 1.5291 (see formula 5)?

Ans. 28.4258 pounds.

4. Find the weight of 500 cubic feet of marsh-gas at a temperature of 60° F. and a pressure due to 30 inches of barometer, the gas having a specific gravity of 0.559. Ans. 21.41 pounds, nearly.

---

833. Matter.—Matter is the substance of which the universe consists. It is indestructible and subject to changes of form under different conditions of heat and pressure; therefore, we find it assuming all the three forms common to matter; namely, the gaseous, liquid, and solid. In all the conditions in which we find matter, it consists of atoms and molecules.

834. Atoms and Molecules.—*Atoms.*—An atom is the smallest conceivable division of matter; and, hence, an atom is always simple in its character.

Molecules.—A molecule is formed by the chemical union of two or more atoms. The atoms composing a molecule may be *like* or *unlike;* and, hence, the molecule may be either simple or compound.

The force that binds *atoms* together to form a molecule is a chemical force, which we call *affinity.*

The *force* that binds *molecules* together to form *mass* is a molecular force, which we call *attraction.*

Affinity binds *atoms* together.

Attraction unites *molecules.*

835. Elements.—An elementary body consists of a simple substance that can not be analyzed or reduced to parts that have other properties than those peculiar to itself. An element is a substance, or form of matter, composed wholly of *like* atoms. Thus, *hydrogen* is an element, because it is composed only of hydrogen atoms. For the same reason, oxygen, nitrogen, carbon, iron, lead, silver, gold, etc., are all elements. Table 17 comprises the most of the elements now known.

836. Compounds.—Any substance or form of matter that is composed of *unlike* atoms is a compound. Two classes of compounds exist, viz.:

(*a*) **Chemical compounds,** in which the combining atoms unite in definite, fixed proportions, according to chemical laws, which give to the atoms of each element certain combining powers. For example, water is a chemical compound, being always formed by the union of *two* atoms of hydrogen to *one* atom of oxygen.

In like manner, when *one* atom of carbon unites with *one* atom of oxygen, carbonic oxide gas is formed; but when *one* atom of carbon unites with *two* atoms of oxygen, carbonic acid gas is produced. These two gases have very different properties.

Again, when *one* atom of carbon unites with *four* atoms of hydrogen, marsh-gas results; but when *two* atoms of carbon unite with the *four* atoms of hydrogen, olefiant gas (ethene) is produced.

These are all examples of chemical compounds, as are also salt, blue vitriol, nitric acid, etc., for they are all formed by the chemical union of dissimilar atoms.

(*b*) **Mechanical mixtures** are not *true compounds*, as they are composed more properly of *unlike molecules*, in place of *unlike atoms.* The molecules of the different substances forming the mixture may be present in any proportions, and the mixture will have properties varying with the proportions of the ingredients.

The atmosphere about us is a good example of a mechanical mixture, as we shall see later, for it consists principally of oxygen and nitrogen gases, mixed in a free state (having no chemical bond of union). The proportion of these two gases in the atmosphere is quite constant, being approximately *one* of oxygen to *four* of nitrogen.

Solutions of different salts in water are examples of mechanical mixtures; the strength of the solution or mixture will vary with the amount of salt dissolved.

TABLE 17.

Name.	Symbol.	Atomic Weight.	Name.	Symbol.	Atomic Weight.
* **Aluminium,**	*Al.*	27.4	**Mercury,**	*Hg.*	200.0
* Antimony,	*Sb.*	122.0	* Molybdenum,	*Mo.*	96.0
* Arsenic,	*As.*	75.0	Nickel,	*Ni.*	58.0
Barium,	*Ba.*	137.0	Niobium (Col-		
Beryllium (Glu-			umbium, Cb.),	*Nb.*	94.0
cinum, Gl.),	*Be.*	9.2	***Nitrogen,***	*N.*	14.0
Bismuth,	*Bi.*	210.0	Osmium,	*Os.*	200.0
Boron,	*B.*	11.0	***Oxygen,***	*O.*	16.0
Bromine,	*Br.*	80.0	Palladium,	*Pd.*	106.0
Cadmium,	*Cd.*	112.0	* *Phosphorus,*	*P.*	31.0
Cæsium,	*Cs.*	133.0	Platinum,	*Pt.*	197.4
Calcium,	*Ca.*	40.0	**Potassium,**	*K.*	39.1
* ***Carbon,***	*C.*	12.0	Rhodium,	*Ro.*	104.0
Cerium,	*Ce.*	91.3	Rubidium,	*Rb.*	85.4
Chlorine,	*Cl.*	35.5	Rüthenium,	*Ru.*	104.0
* Chromium,	*Cr.*	52.2	Samarium,	*Sm.*	150.0
Cobalt,	*Co.*	60.0	Scandium,	*Sc.*	44.9
* Columbium (Ni-			* Selenium,	*Se.*	79.0
obium, Nb.),	*Cb.*	94.0	* ***Silicon,***	*Si.*	28.0
Copper,	*Cu.*	63.4	**Silver,**	*Ag.*	108.0
Decipium,	*Dp.*	159.0	**Sodium,**	*Na.*	23.0
Didymium,	*D.*	95.0	Strontium,	*Sr.*	88.0
Erbium,	*E.*	112.6	* ***Sulphur,***	*S.*	32.0
Fluorine,	*F.*	19.0	* Tantalum,	*Ta.*	182.0
Gallium,	*Ga.*	69.8	* Tellurium,	*Te.*	128.0
Glucinum (Beryl-			Terbium,	*Tb.*	75.4
lium, Be.),	*Gl.*	9.2	Thallium,	*Tl.*	204.0
* **Gold,**	*Au.*	197.0	Thorium,	*Th.*	118.4
* **Hydrogen,**	*H.*	1.0	* Tin,	*Sn.*	118.0
Indium,	*In.*	113.4	* Titanium,	*Ti.*	50.0
Iodine,	*I.*	127.0	* Tungsten,	*W.*	184.0
Iridium,	*Ir.*	198.0	* Uranium,	*U.*	120.0
Iron,	*Fe.*	56.0	Vanadium,	*V.*	51.3
Lanthanum,	*La.*	92.0	Ytterbium,	*Yb.*	173.0
Lead,	*Pb.*	207.0	Yttrium,	*Y.*	89.0
Lithium,	*Li.*	7.0	Zinc,	*Zn.*	65.0
Magnesium,	*Mg.*	24.0	Zirconium,	*Zr.*	89.6
* Manganese,	*Mn.*	55.0			

* Sometimes basic, sometimes acid.

NOTE.—Heavy-faced type indicates the elements of most importance to the student of this subject. Basic elements are printed in common type; acid elements in italics.

837. Dissociation.—This part of our subject would not be complete without some particular reference to the mode of union and disunion of atoms. The affinities which exist between the atoms of the different elements vary very much. In some cases, the affinity is so slight as to render the compound very unstable, and dissociation of the atoms will ensue from the least cause. Examples of this are the various fulminators and some of the detonating explosives. On the other hand, the affinities between atoms of certain other elements, as oxygen and hydrogen, or oxygen and carbon, are very strong, and their union is very apt to be accompanied with the manifestation of considerable energy, either in the form of heat or mechanical work.

In this development of energy, incident to the dissociation of atoms, lies one of the most important principles in the chemistry of mining.

838. Atomic Weights.—By this we mean the weights of the atoms of all the elementary bodies. These weights are expressed in terms of the weight of the hydrogen atom; that is, the lightest known element in nature; for example, it is said the atomic weight of nitrogen is 14, meaning to say that an atom of nitrogen is 14 times as heavy as an atom of hydrogen, and so on with the atomic weights of the other elements.

Table 17 gives the most of the known elements, with their respective symbols and atomic weights.

Atomic weight means only *relative weight.* It does not mean pounds, or ounces, or grammes, or any other denomination in particular. For example, since, from analysis, we know that water is a chemical compound formed by the union of *two* atoms of hydrogen with *one* atom of oxygen, the **molecular weight** (weight of 1 molecule) of water (H_2O) will be as follows:

Hydrogen (H_2), 2 atoms (2×1) $= \ \ 2$
Oxygen (O), 1 atom $= 16$

Water (H_2O), 1 molecule $= 18 =$ molecular weight.

Then, we readily see that hydrogen forms $\frac{2}{18} = \frac{1}{9}$ of the

weight of water, and oxygen forms $\frac{16}{18} = \frac{8}{9}$ of the weight of the same.

In the same way, the atomic weight of carbon being 12 and that of hydrogen 1, we have for the weight of a molecule (called *molecular weight*) of marsh-gas (CH_4):

Carbon (C), 1 atom	$= 12$	
Hydrogen (H_4), 4 atoms (4×1)	$= 4$	
Marsh-gas (CH_4), 1 molecule	$= 16$	= molecular weight.

We see, therefore, that the carbon forms $\frac{12}{16} = \frac{3}{4}$, or 75 per cent. by weight of the marsh-gas, and hydrogen forms $\frac{4}{16} = \frac{1}{4}$, or 25 per cent. of the same.

This makes plain that the atomic weights of the various elements are only relative, hydrogen being taken as unity.

EXAMPLES.—(*a*) What per cent. of the weight of carbonic oxide gas is pure oxygen?

SOLUTION.—Carbonic oxide gas (CO) contains one atom of carbon and one atom of oxygen, by weight, carbon 12 parts and oxygen 16 parts (see Table 17), 12 and 16 being the relative weights of the atoms, or the atomic weights, of carbon and oxygen. The molecular weight, therefore, of carbonic oxide gas is $12 + 16 = 28$, of which oxygen forms $\frac{16}{28} = \frac{4}{7} = 57.143$ per cent., nearly. Ans.

(*b*) What weight of carbonic acid gas will be produced in burning 100 pounds of coal containing 90 per cent. of carbon?

SOLUTION.— 90% of 100 = 90 pounds of carbon.

Carbon (C) atomic weight (Table 17) 1 atom	12
Oxygen (O_2) atomic weight (Table 17) 2 atoms (16×2)	32
Carbonic acid gas (CO_2), molecular weight	44

Percentage of carbon $= \frac{12}{44} = \frac{3}{11} = 27.27\%$. Then, 90 pounds of carbon is $\frac{3}{11}$ of the weight of carbonic acid gas formed, and $\frac{1}{11} = \frac{1}{3}$ of $90 = 30$ pounds, and $\frac{11}{11}$ or the whole weight of gas formed $= 30 \times 11 = 330$ pounds. Ans.

(*c*) If the specific gravity of carbonic acid gas is 1.5291, what volume of gas, at a temperature of 60° F., and a barometric pressure equal to 30″, will be produced, in the last problem, by burning the 100 pounds of coal, which we found yields 330 pounds of this gas?

SOLUTION.—The specific gravity of the gas being 1.5291, one cubic foot at the temperature and pressure given will weigh $.0766 \times 1.5291 = .117129$ pounds; and 330 pounds of the gas will contain as many cubic feet as $\frac{330}{.117129} = 2{,}817.4$ cubic feet. Ans.

EXAMPLES FOR PRACTICE.

1. What percentage of the weight of marsh-gas (CH_4) is pure hydrogen (a molecule of marsh-gas containing *one* atom of carbon and *four* atoms of hydrogen)? Ans. 25%.

2. If the specific gravity of marsh-gas (Table 19) is 0.559, what weight of hydrogen will be contained in 1,000 cubic feet of this gas, at a temperature of 60° F., barometer 30"? Ans. 10.705 lb., nearly.

3. If all of the hydrogen in example 2 were to unite with oxygen (in the proportion of *two* atoms of hydrogen to *one* atom of oxygen) to form water, what weight of water would be produced? Ans. 96.345 lb.

4. What weight of carbon is contained in 100 cubic feet of carbonic oxide gas (CO), a molecule of this gas containing *one* atom of carbon and *one* atom of oxygen, temperature = 60° F., barometer 30"? Ans. 3.1745 lb.

---

839. Symbols.—To facilitate the writing of chemical equations, each of the elements is expressed in writing by a symbol, as given in Table 17. In like manner, a chemical compound is expressed by the symbols of its constituent elements. Thus, water, composed of *two* atoms of hydrogen and *one* atom of oxygen, is expressed by the symbol H_2O. In the same manner, we write CO for carbonic oxide gas, CO_2 for carbonic acid gas, and CH_4 for carbureted hydrogen or marsh-gas; the number of atoms of each element, when more than one, being denoted by the little subscript figure following its symbol. When it is desired to express more than one *molecule*, we write the figure indicating the number before the formula of the molecule. Thus, four molecules of carbonic acid gas are written $4\,CO_2$, and in these four molecules there are *four* atoms of carbon and *eight* atoms of oxygen.

These symbols are, in most cases, the first letter of the name of the element itself. For example, the symbol for oxygen is O, for hydrogen it is H; and such symbols are always capital letters. When the initial letters of the names of different elements are the same, one of the elements is designated by its initial letter; but each of the others has some additional letter to distinguish it, and this extra letter, in each case, is a small letter. Thus, the symbol for carbon

is C; for chlorine, Cl; for chromium, Cr; for cadmium, Cd, etc. Others of the elements have for their symbols an abbreviation of their Latin names; as, for example, the symbol for iron is Fe (Latin, *ferrum*), and silver, Ag (Latin, *argentum*). In writing chemical symbols, be careful that you do not use capitals and small letters indiscriminately, for the meaning may thereby be greatly changed. Thus, were we to write CO (carbonic oxide), a deadly gas would be meant, while if it had been made Co, it would represent the chemical symbol for cobalt.

840. Chemical Equations.—An equation expresses equality. We may have a numerical equation; as, for example,

$$2 + 4 = 3 \times 2.$$

The sign of equality divides the two equal members of an equation, always showing that they are equal to each other.

A *chemical equation* is used to show the arrangement and grouping of atoms, *before* and *after* a reaction. We must remember that *matter may be changed in its form, but can not be destroyed;* hence, the grouping of the atoms will be different after the reaction from what it was before the reaction took place; but the number and kind of atoms will be equal before and after such reaction. For example, when sulphuric acid (H_2SO_4) acts upon metallic zinc (Zn), the hydrogen of the acid is replaced by the zinc, and the result is that a salt (sulphate of zinc) is formed and a gas (hydrogen) is set free. This reaction is expressed by the following chemical equation:

$$Zn + H_2SO_4 = ZnSO_4 + 2H.$$

841. Atomic volume is relative volume, as atomic weight is relative weight. It has been ascertained that, with few exceptions, the specific gravities of simple gases, when referred to hydrogen as unity, are equal to their respective atomic weights; as, for example, the density (specific gravity) of nitrogen, referred to hydrogen, is 14; and, referring to Table 17, we see that its atomic weight is,

likewise, 14. From the foregoing has been deduced the following law:

First Law of Volume.—(*a*) *Like volumes of simple gases contain the same number of atoms.*

(*b*) *The atoms of all simple gases are of the same size.*

It has been further ascertained that the densities of compound gases are, with few exceptions, equal to one-half of their *molecular* weights; as, for example, the density of carbonic acid gas (CO_2), referred to hydrogen as unity, is 22; its molecular weight (sum of the atomic weights of its elements) is $12 + (2 \times 16) = 44$. (See Table 17.) Hence, we see that its density is one-half of its molecular weight. From this the following law has been deduced:

Second Law of Volume.—*The molecules of compound gases occupy twice the volume of an atom of hydrogen gas.*

The exceptions to these general laws of volume are very few, and not important to our subject. The foregoing rules do not refer to solids or liquids.

These two *laws of volume* make it possible to determine the volume of gases resulting from any given chemical reaction. For example, when carbonic oxide gas burns, it unites with the oxygen of the air according to the equation

$$CO + O = CO_2.$$

But the molecule CO (1 atom C and 1 atom O) is, according to the second law of volume, equal to the size of *two* atoms of hydrogen gas; and, according to the first law of volume, the atom O, with which it unites, is equal in size to *one* atom of hydrogen gas. Hence, the volumes of CO and O, which unite, are to each other as 2 : 1. In the same manner, we find the volume of CO_2 formed is equal to the original volume of CO. In other words, *two* volumes of carbonic oxide gas, mixed with *one* volume of oxygen, and exploded, will form only *two* volumes of carbonic acid gas.

In like manner, when ammonia (NH_3) is decomposed in a tube, by electric sparks, it is found that *two* volumes NH_3 yield *one* volume N and *three* volumes H, or *four* volumes of the simple gases.

EXAMPLE.—Determine the volume of carbonic acid gas (CO_2) resulting from the explosion of 500 cubic feet of marsh-gas (CH_4), at equal temperatures and pressures.

SOLUTION.—Assuming that all of the carbon (C) of the marsh-gas is converted into carbonic acid gas (CO_2), we find

1 *molecule* CH_4 *yields* 1 *molecule* CO_2.

By the second law of volume, each of these molecules occupies twice the volume of an atom of hydrogen gas, and they are, therefore, equal to each other. Hence, 500 cubic feet of marsh-gas yield 500 cubic feet of carbonic acid gas, under the assumed conditions (equal temperatures and pressures). Ans.

EXAMPLE.—How many cubic feet of oxygen have been consumed in the formation of the 500 cubic feet of carbonic acid gas of the previous example?

SOLUTION.—*Two atoms* of oxygen are consumed in the formation of each *molecule* of carbonic acid gas (CO_2). These *two* atoms, according to the first law of volume, are of the same size or volume as *two* atoms of *hydrogen* gas; likewise, according to the second law of volume, the *molecule* of carbonic acid gas formed occupies *twice* the volume of an atom of *hydrogen* gas. Hence, the volume of the oxygen consumed is equal to the volume of the gas formed (500 cu. ft.). Ans.

EXAMPLES FOR PRACTICE.

In the following examples, assume constant temperature and pressure:

1. How many cubic feet of oxygen will be consumed in the formation of 100 cubic feet of carbonic oxide gas (CO)? Ans. 50 cu. t.

2. If the hydrogen in 100 cubic feet of ammonia gas were set free, what volume would it make? Ans. 150 cu. ft.

3. The formula for ethene, or olefiant gas, is C_2H_4; what volume of oxygen will be required to convert 100 cubic feet of this gas into CO_2 and H_2O? Ans. 300 cu. ft.

---

842. Constitution of Matter.—In order to rightly understand the relation of force to matter, we must consider the latter as made up of minute particles, which we have already termed atoms and molecules.

The union of atoms produces molecules, and the union of molecules produces mass.

The *atoms* forming the molecules of a substance may be *like* or *unlike*. When they are *like*, the substance is *elementary;* when unlike, it is *compound.*

The *molecules* of any homogeneous mass are always alike.

843. Molecular Forces.—The force which unites *atoms* is *affinity;* it is a chemical force.

The *molecules* of all matter are acted upon by two opposite or contrary forces; viz., the force of *attraction* and the force of *repulsion*. The former of these two forces acts to bind the molecules together, the latter to drive them apart. The *attractive* principle or force exists in every molecule of a mass, to draw it towards every other molecule; it is an inherent force, peculiar to all matter to a greater or less extent.

The *repulsive* force existing between the molecules of a mass is what may be termed an *imposed* force. It is not common to the mass, but is an induced or applied force. This repulsive force is largely the result of heat or the temperature of the mass.

844. Heat Unit.—We measure the quantity of heat by what is termed the *thermal*, or *heat*, *unit*. The British thermal unit is the amount of heat which will raise the temperature of one pound of water one degree of the Fahrenheit scale. Table 18 gives, in round numbers, the number of British thermal units produced by the burning of one pound of different solids and gases in oxygen.

TABLE 18.

Substance.	British Thermal Units (per Pound).
Hydrogen gas (H)	62,000
Marsh-gas (CH_4)	23,500
Carbonic oxide gas (CO)	4,300
Anthracite coal	15,230
Bituminous coal	14,400
Coke	12,600
Wood (ordinary)	5,000

AIR AND GASES.

THE ATMOSPHERE.

845. Composition.—An analysis of the atmosphere about us shows it to consist of a *mixture* of oxygen and nitrogen, with varying amounts of carbonic acid gas and ammonia. The oxygen and nitrogen are always free or uncombined, and are present in the proportions given below.

	By Volume.	By Weight.
Nitrogen,	79.3	77
Oxygen,	20.7	23
	100.0	100

The amounts of the other ingredients are changing all the time, due to local causes. Thus, the air of a crowded room, or a mine, or the air in the vicinity of large factories, may show a high percentage of carbonic acid gas; while, again, the air of an open field, just after a shower, may show scarcely a trace of this gas; while the proportions of oxygen and nitrogen will show no practical variation. The proportions of these two elements existing in the atmosphere, by volume, is in the ratio of about four volumes of nitrogen to one volume of oxygen. The oxygen and nitrogen are in a *free* state; that is, they are mechanically *mixed* in this proportion throughout the entire atmosphere, and are not chemically *combined.*

The atmosphere is essential to all animal and vegetable life. Its oxygen, which forms about one-fifth of its volume, enters into a large number of chemical reactions, and supports combustion in every form.

PHYSICAL PROPERTIES OF AIR AND GASES.

846. Weight of Air.—It was supposed by the ancients that air had no weight, and it was not until about the year 1650 that it was proven that air really has weight. A cubic inch of air, under ordinary conditions, weighs .31 grain, nearly. The ratio of the weight of air to water is about

1 : 774; that is, air is only $\frac{1}{774}$ as heavy as water. If a vessel, made of light material, be filled with a gas lighter than air, so that the total weight of the vessel and gas is less than the air which they displace, the vessel will rise. It is on this principle that balloons are made.

Since air has weight, it is evident that the enormous quantity of air that constitutes the atmosphere must exert a considerable pressure upon the earth. This is easily proven by taking a long glass tube closed at one end and filling it with mercury. If the finger be placed over the open end so as to keep the mercury from running out, and the tube be inverted and placed in a glass of mercury, as shown in Fig. 110, the mercury in the tube will fall, then rise, and, after a few oscillations, will come to rest at a height above the top of the mercury in the glass equal to about 30 inches. This height will always be the same under the same atmospheric conditions. Now, if the atmosphere has weight, it must press upon the upper surface of the mercury in the glass with equal intensity upon every square unit, except upon that part of the surface occupied by the tube. In order that there will be equilibrium, the weight of the mercury in the tube must be equal to the pressure of the air upon an area of the upper surface of the mercury in the glass, equal to the area of the inside of the tube. Suppose that the area of the inside of the tube is 1 square inch, then, since mercury is 13.6 times as heavy as water, and a cubic inch of water weighs .03617 pound, the weight of the mercurial column is $.03617 \times 13.6 \times 30 = 14.7574$ pounds.

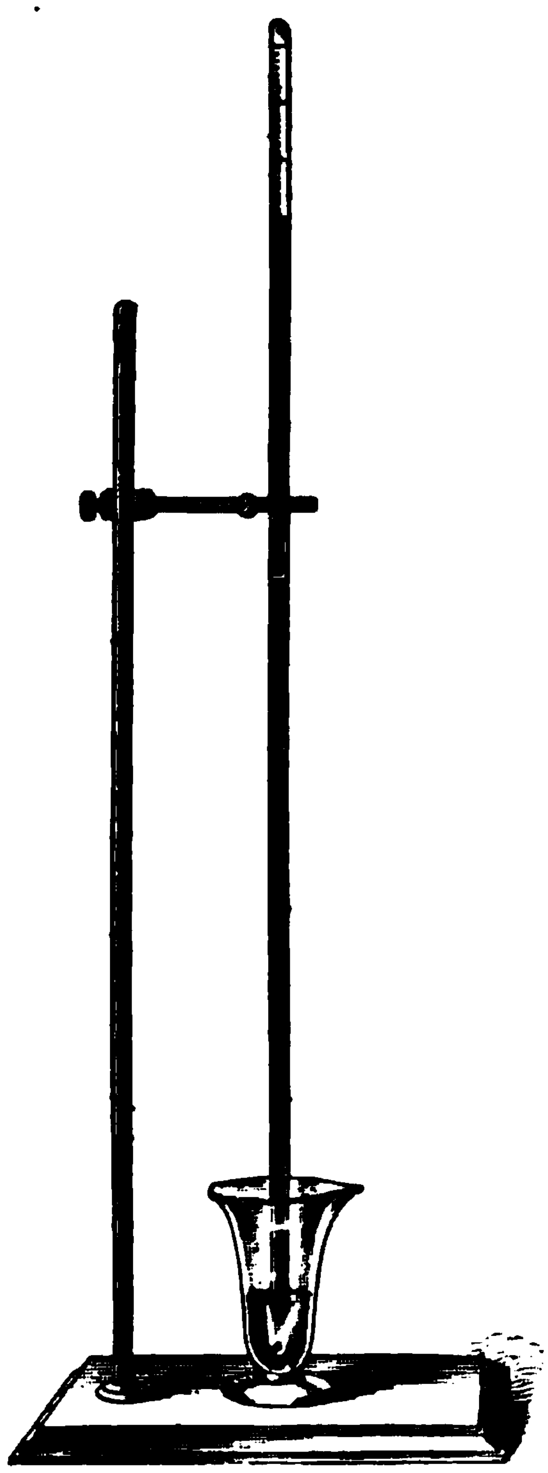


FIG. 110.

The actual height of the mercury is a little less than 30 inches, and the actual weight of a cubic inch of distilled water is a little less than .03617 pound. When these considerations are taken into account, the average weight of the mercurial column at the level of the sea under normal conditions is 14.69 pounds, or, practically, 14.7 pounds. Since this weight, exerted upon 1 square inch of the liquid in the glass, just produced equilibrium, it is plain that the pressure of the outside air is 14.7 pounds upon every square inch of surface.

847. Vacuum.—The space between the upper end of the tube and the upper surface of the mercury is called a *Toricellian vacuum*, or simply a *vacuum*, meaning that it is an entirely empty space, and does not contain any substance, solid, liquid, or gaseous. If there was a gas of some kind there, no matter how small the quantity might be, it would expand, filling the space, and its tension would cause the column of mercury to fall and become shorter, according to the amount of gas or air present. The space is then called a *partial vacuum*. If the mercury fell 1 inch, so that the column was only 29 inches high, we would say, in ordinary language, that there were *29 inches of vacuum*. If it fell 8 inches, we would say that there were 22 inches of vacuum; if it fell 16 inches, we would say that there were 14 inches of vacuum, etc. Hence, when the vacuum-gauge of a condensing-engine shows 26 inches of vacuum, there is enough air in the condenser to produce a pressure of $\frac{30 - 26}{30} \times 14.7 = \frac{4}{30} \times 14.7 = 1.96$ pounds per square inch.

If the tube had been filled with water instead of mercury, the height of the column of water to balance the pressure of the atmosphere would have been $30 \times 13.6 = 408$ inches = 34 feet. This means that if a tube be filled with water, inverted, and placed in a dish of water in a manner similar to the experiment made with the mercury, the height of the column of water would be 34 feet.

848. The **barometer** is an instrument used for measuring the pressure of the atmosphere. There are two kinds in general use—the mercurial barometer and the aneroid barometer. The *mercurial barometer* is shown in Fig. 111. The principle is the same as the inverted tube, shown in Fig. 110. In this case, the tube and cup at the bottom are protected by a brass or iron casing. Near the top of the tube is a graduated scale which can be read to $\frac{1}{1000}$ of an inch by means of a vernier. Attached to the casing is an accurate thermometer for determining the temperature of the outside air at the time the barometric observation is taken. This is necessary, since mercury expands when the temperature is increased, and contracts when the temperature falls; for this reason a standard temperature is assumed, and all barometer readings are reduced to this temperature. This standard temperature is usually taken at 32° F., at which temperature the average height of the mercurial column at sea-level is 30 inches. Another correction is made for the altitude of the place above sea-level, and a third correction for the effects of capillary attraction.

In Fig. 112 is shown a cut of an *aneroid barometer*. These instruments are made in various sizes, from the size of a watch up to an 8 or 10 inch face. They consist of a cylindrical box of metal with a top of thin, elastic, corrugated metal. The air is removed from the box. When the atmospheric pressure increases, the top is pressed inwards, and when it is diminished, the top is pressed outwards by its own elasticity, aided by a spring beneath. These movements of the cover are transmitted and multiplied by a combination of delicate levers, which act upon an index-hand, and cause it to move either to the right or left, over a graduated scale. These barometers are self-correcting (compensated) for variations in temperature. They are

FIG. 111.

very portable, occupying but a small space, and are so delicate that they are said to show a difference in the atmospheric pressure when transferred from the table to the floor. The mercurial barometer is the standard. With air, as with water, the lower we get, the greater the pressure, and the higher we get, the less the pressure. At the level of the sea, the height of the mercurial column is about 30 inches; at 5,000 feet above the sea, it is 24.7 inches; at

FIG. 112.

10,000 feet above the sea, it is 20.5 inches; at 15,000 feet, it is 16.9 inches; at 3 miles, it is 16.4 inches, and at 6 miles above the sea-level, it is 8.9 inches.

849. Density of Air.—The density and weight of a cubic foot of air vary with the altitude; that is, a cubic foot of air at an elevation of 5,000 feet above the

sea-level will not weigh as much as a cubic foot at sea-level. This is proven conclusively by the fact that at a height of 3½ miles the mercurial column measures but 15 inches, indicating that half the weight of the entire atmosphere is below that. It is known that the height of the earth's atmosphere is at least 50 miles; hence, the air just before reaching the limit must be in an exceedingly rarefied state. It is by means of barometers that great heights are measured. The aneroid barometer has the heights marked on the dial, so that they can be read directly. With the mercurial barometer, the heights must be calculated from the reading.

850. Atmospheric Pressure.—The atmospheric pressure is everywhere present, and presses all objects in all directions with equal force. If a book is laid upon the table, the air presses upon it in every direction with an equal average force of 14.7 pounds per square inch. It would seem as though it would take considerable force to raise a book from the table, since, if the size of the book were 8 inches by 5 inches, the pressure upon it is $8 \times 5 \times 14.7 = 588$ pounds; but there is an equal pressure beneath the book to counteract the pressure on the top. It would now seem as though it would require a great force to open the book, since there are two pressures of 588 pounds each, acting in opposite directions, and tending to crush the book; so it would but for the fact that there is a layer of air between each leaf acting upwards and downwards with a pressure of 14.7 pounds per square inch. If two metal plates be made as perfectly smooth and flat as it is possible to get them, and the edge of one be laid upon the edge of the other, so that one may be slid upon the other, and thus exclude the air, it will take an immense force, compared with the weight of the plates, to separate them. This is because the full pressure of 14.7 pounds per square inch is then exerted upon each plate, with no counteracting equal pressure between them.

If a piece of flat glass be laid upon a flat surface that has been previously moistened with water, it will require considerable force to separate them; this is because the water

excludes the air between the flat surface and glass, and any attempt to separate these causes a partial vacuum between the glass and the surface, thereby reducing the counter pressure beneath the glass.

851. Tension of Gases.—In Fig. 110 the space above the column of mercury was said to be a vacuum, and that if any gas or air was present it would expand, its tension forcing the column of mercury downwards. If enough gas is admitted to cause the mercury to stand at 15 inches, the tension of the gas is evidently $\frac{14.7}{2} = 7.35$ pounds per square inch, since the pressure of the outside air of 14.7 pounds per square inch balances only 15 inches instead of 30 inches of mercury; that is, it balances only half as much as it would if there were no gas in the tube; therefore, the tension (pressure) of the gas in the tube is 7.35 pounds. If more gas is admitted, until the top of the mercurial column is just level with the mercury in the cup, the gas in the tube has then a tension equal to the outside pressure of the atmosphere. Suppose that the bottom of the tube is fitted with a piston, and that the total length of the inside of the tube is 36 inches. If the piston be shoved upwards so that the space occupied by the gas is 18 inches long instead of 36 inches, the temperature remaining the same as before, it will be found that the tension of the gas within the tube is 29.4 pounds. It will be noticed that the volume occupied by the gas is only half that in the tube before the piston was moved, while the pressure is twice as great, since $14.7 \times 2 = 29.4$ pounds. If the piston be shoved up, so that the space occupied by the gas is only 9 inches instead of 18 inches, the temperature still remaining the same, the pressure will be found to be 58.8 pounds per square inch. The volume has again been reduced one-half, and the pressure increased two times, since $29.4 \times 2 = 58.8$ pounds. The volume now occupied by the gas is 9 inches long, whereas, before the piston was moved, it was 36 inches long; as the tube was assumed to be of uniform diameter throughout its length, the volume is now $\frac{9}{36} = \frac{1}{4}$ of its original volume,

and its pressure is $\frac{58.8}{14.7} = 4$ times its original pressure. Moreover, if the temperature of the confined gas remains the same, the pressure and volume will always vary in a similar way. The law which states these effects is called *Mariotte's law.*

852. Mariotte's Law.—*The temperature remaining the same, the volume of a given quantity of gas varies inversely as the pressure.*

The meaning of this is: If the volume of the gas is diminished to $\frac{1}{2}$, $\frac{1}{3}$, $\frac{1}{5}$, etc., of its former volume, the tension will be increased 2, 3, 5, etc., times, or, if the outside pressure be increased 2, 3, 5, etc., times, the volume of the gas will be diminished to $\frac{1}{2}$, $\frac{1}{3}$, $\frac{1}{5}$, etc., of its original volume, the temperature remaining constant. It also means that if a gas is under a certain pressure, and the pressure is diminished to $\frac{1}{2}$, $\frac{1}{4}$, $\frac{1}{10}$, etc., of its original pressure, that the volume of the confined gas will be increased 2, 4, 10, etc., times—its tension decreasing at the same rate.

Suppose 3 cubic feet of air to be under a pressure of 60 pounds per square inch in a cylinder fitted with a movable piston; then the product of the volume and pressure is $3 \times 60 = 180$. Let the volume be increased to 6 cubic feet, then the pressure will be 30 pounds per square inch, and $30 \times 6 = 180$ as before. Let the volume be increased to 24 cubic feet, it is then $\frac{24}{3} = 8$ times its original volume, and the pressure is $\frac{1}{8}$ of its original pressure, or $60 \times \frac{1}{8} = 7\frac{1}{2}$ pounds, and $24 \times 7\frac{1}{2} = 180$, as in the two preceding cases. It will now be noticed that if a gas be enclosed within a confined space, and allowed to expand without losing any heat, *the product of the pressure and the corresponding volume for one position of the piston is the same as for any other position of the piston.* If the piston was to compress the air, the rule would still hold good.

Let p = pressure for one position of the piston;
p_1 = pressure for any other position of the piston;
v = volume corresponding to the pressure p;
v_1 = volume corresponding to the pressure p_1.

Then, $$pv = p_1 v_1; \qquad (6.)$$

also, $$p_1 = \frac{pv}{v_1}; \qquad (7.)$$

and $$v_1 = \frac{pv}{p_1}. \qquad (8.)$$

Knowing the volume and the pressure for any position of the piston and the volume for any other position, the pressure may be calculated by formula **7**, or if the pressure is known for any other position, the volume may be calculated by formula **8**.

EXAMPLE.—If 1.875 cubic feet of air be under a pressure of 72 pounds per square inch, (*a*) what will be the pressure when the volume is increased to 2 cubic feet? (*b*) to 3 cubic feet? (*c*) to 9 cubic feet?

SOLUTION.—(*a*) $p_1 = \frac{pv}{v_1} = \frac{72 \times 1.875}{2} = 67\frac{1}{2}$ pounds per square inch. Ans.

(*b*) $p_1 = \frac{72 \times 1.875}{3} = 45$ pounds per square inch. Ans.

(*c*) $p_1 = \frac{72 \times 1.875}{9} = 15$ pounds per square inch. Ans.

EXAMPLE.—If 10 cubic feet of air have a tension of 5.6 pounds per square inch, (*a*) what is the volume when the tension is 4 pounds? (*b*) 8 pounds? (*c*) 25 pounds? (*d*) 100 pounds?

SOLUTION.—(*a*) $v_1 = \frac{pv}{p_1} = \frac{5.6 \times 10}{4} = 14$ cubic feet. Ans.

(*b*) $v_1 = \frac{5.6 \times 10}{8} = 7$ cubic feet. Ans.

(*c*) $v_1 = \frac{5.6 \times 10}{25} = 2.24$ cubic feet. Ans.

(*d*) $v_1 = \frac{5.6 \times 10}{100} = .56$ cubic foot. Ans.

As a necessary consequence of Mariotte's law, it may be stated that *the density of a gas varies directly as the pressure and inversely as the volume;* that is, *the density increases as the pressure increases, and decreases as the volume increases.*

This is evident, since if a gas has a tension of two atmospheres, or $14.7 \times 2 = 29.4$ pounds per square inch, it will weigh twice as much as the same volume would if the

tension was one atmosphere, or 14.7 pounds per square inch. For, let the volume be increased until it is twice as great as the original volume, the tension will then be one atmosphere. The total weight of the gas has not been changed, but there are now 2 cubic feet for every 1 cubic foot of the original volume, and the weight of one cubic foot now is only half as great as before. Thus, the density decreases as the volume increases, and as an increase of pressure causes a decrease of volume, the density increases as the pressure increases.

Let D be the density corresponding to the pressure p and volume v, and D_1 be the density corresponding to the pressure p_1 and volume v_1; then,

$$p : D :: p_1 : D_1, \text{ or } p D_1 = p_1 D, \qquad (9.)$$

and $$v : D_1 :: v_1 : D, \text{ or } v D = v_1 D_1. \qquad (10.)$$

Since the weight is proportional to the density, the weights may be used in place of the densities in formulas **9** and **10.** Thus, let W be the weight of a quantity of air or other gas whose volume is v and pressure is p; let W_1 be the weight of the same quantity when the volume is v_1 and pressure is p_1; then,

$$p : W :: p_1 : W_1, \text{ or } p W_1 = p_1 W, \qquad (11.)$$

$$v : W_1 :: v_1 : W, \text{ or } v W = v_1 W_1. \qquad (12.)$$

EXAMPLE.—The weight of 1 cubic foot of air at a temperature of 60° F. and under a pressure of 1 atmosphere (14.7 pounds per square inch) is .0763 pound; what would be the weight per cubic foot if the volume was compressed until the tension was 5 atmospheres, the temperature still being 60°?

SOLUTION.—Using formula **11**,

$p : W :: p_1 : W_1$, or $1 : .0763 :: 5 : W_1$, or $W_1 = .3815$ lb. Ans.

EXAMPLE.—If in the last example the air had expanded until the tension was 5 pounds per square inch, what would have been its weight per cubic foot?

SOLUTION.—Here $p = 14.7$, $p_1 = 5$, and $W = .0763$. Hence, using the same formula, $14.7 : .0763 :: 5 : W_1$, or $W_1 = .02595$ lb. Ans.

EXAMPLE.—If 6.75 cubic feet of air at a temperature of 60° F., and a pressure of one atmosphere, are compressed to 2.25 cubic feet (the temperature still remaining 60° F.), what is the weight of a cubic foot of the compressed air?

SOLUTION.—Using formula **12**,

$$v : W_1 :: v_1 : W, \text{ or } 6.75 : W_1 :: 2.25 : .0763,$$

or
$$W_1 = \frac{.0763 \times 6.75}{2.25} = .2289 \text{ lb. Ans.}$$

853. Relation of Temperature to Volume.—In all that has been said before, it has been stated that the temperature was constant; the reason for this will now be explained: Suppose 5 cubic feet of air to be confined in a cylinder whose area is 10 square inches, placed in a vacuum so that there will be no pressure due to the atmosphere, and the cylinder be fitted with a piston weighing, say, 100 pounds. The tension of the gas will be $\frac{100}{10} = 10$ pounds per square inch. Suppose that the temperature of the air is 32° F., and that it is heated until the temperature is 33° F., or the temperature is increased 1°; it will be found that the piston has risen a certain amount, and, consequently, the volume has increased, while the pressure is the same as before, or 10 pounds per square inch. If more heat is applied until the temperature of the gas is 34° F., it will be found that the piston has again risen and the volume again increased, while the pressure still remains the same. It will be found that for every increase of temperature, there will be a corresponding increase of volume. The law which expresses this change is called *Gay-Lussac's law.*

854. Gay-Lussac's Law.—*If the pressure remains constant, every increase of temperature of 1° F. produces in a given quantity of gas an expansion of $\frac{1}{491}$ of its volume at 32° F.*

If the pressure remains constant, it will also be found that every decrease of temperature of 1° F. will cause a decrease of $\frac{1}{491}$ of the volume at 32° F.

Let v = volume of gas before heating;
v_1 = volume of gas after heating;
t = temperature corresponding to volume v;
t_1 = temperature corresponding to volume v_1.

Then,
$$v_1 = v\left(\frac{459 + t_1}{459 + t}\right). \qquad \textbf{(13.)}$$

That is, *the volume of gas after heating (or cooling) equals the original volume multiplied by 459 plus the final temperature, divided by 459 plus the original temperature.*

EXAMPLE.—When 5 cubic feet of air at a temperature of 45° are heated under constant pressure up to 177°, what is its new volume?

SOLUTION.—Applying formula **13**,

$$v_1 = v\left(\frac{459 + t_1}{459 + t}\right) = 5 \times \left(\frac{636}{504}\right) = 6.309 \text{ cu. ft.}$$

Suppose that a certain volume of gas is confined in a vessel so that it can not expand; in other words, suppose that the piston of the cylinder before mentioned to be fastened so that it can not move. Let a gauge be placed on the cylinder so that the tension of the confined gas can be registered. If the gas is heated, it will be found that for every increase of temperature of 1° F. there will be a corresponding increase of $\frac{1}{491}$ of the tension at 32° F.; that is, the volume remaining constant, the tension increases $\frac{1}{491}$ of the tension at 32° F. for every degree rise of temperature.

Let p = the original tension;
t = the corresponding temperature;
t_1 = any higher temperature;
p_1 = corresponding tension.

Then, $$p_1 = p\left(\frac{459 + t_1}{459 + t}\right). \qquad \textbf{(14.)}$$

That is, *if a certain quantity of gas is heated from $t°$ to $t_1°$, the volume remaining constant, the resulting tension p_1 will be equal to the original tension multiplied by 459 plus the final temperature, divided by 459 plus the original temperature.*

EXAMPLE.—If a certain quantity of air is heated under constant volume from 45° to 177°, what is the resulting tension, the original tension being 14.7 pounds per square inch?

SOLUTION.—Using formula **14**,

$$p_1 = p\left(\frac{459 + t_1}{459 + t}\right) = 14.7 \times \left(\frac{636}{504}\right) = 18.55 \text{ lb. per sq. in.}$$

855. Absolute Zero.—According to the modern and now generally accepted theory of heat, the atoms and

molecules of all bodies are in an incessant state of vibration. The vibratory movement in the liquids is faster than in the solids; it is faster in the gases than in either of the others. Any increase of heat increases the vibrations, and a decrease of heat decreases them. From experiments and calculations based upon higher mathematics, it has been concluded that at 459° below zero on the Fahrenheit scale, or at 273° below zero on the Centigrade scale, all these vibrations cease. This point is called the *absolute zero*, and all temperatures reckoned from this point are called the *absolute temperatures*. The point of absolute zero has never been reached nor closely approached, the lowest recorded temperature being 360° F. below zero, but, nevertheless, it has a meaning, and is used in many formulas, being nearly always denoted by T. Ordinary temperatures are denoted by t. When the word temperature alone is used, the meaning is the same as ordinarily used, but when absolute temperature is specified, 459° F. must be added to the temperature. The absolute temperature corresponding to 212° F. is 459° + 212° = 671° F. If the absolute temperature is given, the ordinary temperature may be found by subtracting 459° from the absolute temperature. Thus, if the absolute temperature is 520° F., the temperature is 520° − 459° = 61° F.

Let $P =$ pressure per square inch;
$V =$ volume of air in cubic feet;
$T =$ absolute temperature;
$W =$ weight.

Then,
$$P = \frac{.37052\,W\,T}{V}; \qquad (15.)$$

$$V = \frac{.37052\,W\,T}{P}; \qquad (16.)$$

$$T = \frac{P\,V}{.37052\,W}; \qquad (17.)$$

$$W = \frac{P\,V}{.37052\,T}. \qquad (18.)$$

NOTE.—The constant .37052 is the reciprocal of the weight, in pounds, of 1 cubic foot of air at 1° absolute temperature (Fahr.), and a pressure of 1 pound per square inch.

EXAMPLE.—If 40 cubic feet of air weigh 3.5 pounds, and have a temperature of 82°, what is the pressure (tension) in pounds per square inch?

SOLUTION.— $P = \frac{.37052\,W\,T}{V} = \frac{.37052 \times 3.5 \times 541}{40} = 17.539$ lb. per sq. in. Ans.

EXAMPLE.—What is the volume in cubic feet of a certain quantity of air having a tension of 17.539 pounds per square inch, a temperature of 80°, and which weighs 3.5 pounds?

SOLUTION.— $V = \frac{.37052\,W\,T}{P} = \frac{.37052 \times 3.5 \times 541}{17.539} = 40$ cu. ft. Ans.

EXAMPLE.—If 40 cubic feet of air having a tension of 17.539 pounds per square inch weigh 3.5 pounds, what is the temperature?

SOLUTION.— $T = \frac{P\,V}{.37052\,W} = \frac{17.539 \times 40}{.37052 \times 3.5} = 541°$, nearly. Hence, $541° - 459° = 82°$. Ans.

EXAMPLE.—If 40 cubic feet of air have a tension of 17.539 pounds per square inch, and a temperature of 82°, (*a*) what is its weight? (*b*) what is its weight per cubic foot?

SOLUTION.—(*a*) $W = \frac{P\,V}{.37052\,T} = \frac{17.539 \times 40}{.37052 \times 541} = 3.5$ lb. Ans.

(*b*) $3.5 \div 40 = .0875$ lb. per cu. ft. Ans.

856. Mixing of Gases.—If two liquids which do not act chemically upon each other are mixed together and allowed to stand, it will be found that after a time the two liquids have separated, and the heavier has fallen to the bottom. If two vessels containing gases of different densities be put in communication with each other, the gases will mingle freely together till the mixture is uniform in each vessel. If one vessel be above the other, and the heavier gas be in the lower vessel, the same result will occur. The greater the difference of the densities of the gases, the quicker a uniform mixture will be formed, assuming that no chemical action takes place between the gases. When the gases have the same temperature and pressure, the pressure of the mixture will be the same; this is evident, since the total volume has not been changed, and unless the volume or temperature changes, the pressure can not change. This property of the mixing of gases is a very valuable one, since, if they acted like liquids, carbonic acid gas (the result of

combustion), which is $1\frac{1}{2}$ times as heavy as air, would remain next to the earth, instead of dispersing into the atmosphere, the result being that no animal life could exist.

Mixtures of Equal Volumes of Gases Having Unequal Pressures.—*If two gases having the same volume and temperature, but different pressures, be mixed in a vessel whose volume equals one of the equal volumes of the gas, the pressure of the mixture will be equal to the sum of the two pressures, provided that the temperature remains the same as before.*

EXAMPLE.—Two vessels containing 3 cubic feet of gas, each at a temperature of 60°, and at a pressure of 40 pounds and 25 pounds per square inch, respectively, are placed in communication with each other, and all the gas is compressed into one vessel. If the temperature of the mixture is also 60°, what is the pressure?

SOLUTION.—According to the law just given, the pressure will be $40 + 25 = 65$ lb. per sq. in.

857. Mixture of Two Gases Having Unequal Volumes and Pressures.—

Let v and p be the volume and pressure of one of the gases.
Let v_1 and p_1 be the volume and pressure of the other gas.
Let V and P be the volume and pressure of the mixture.

Then, if the temperature remains the same,

$$P = \frac{p\,v + p_1\,v_1}{V}. \qquad \textbf{(19.)}$$

$$V = \frac{p\,v + p_1\,v_1}{P}. \qquad \textbf{(20.)}$$

EXAMPLE.—Two gases of the same temperature, having volumes of 7 cubic feet and $4\frac{1}{2}$ cubic feet, and tensions of 25 pounds and 18 pounds per square inch, respectively, are mixed together in a vessel whose volume is 10 cubic feet. The temperature remaining the same, what is the resulting pressure?

SOLUTION.— $P = \frac{p\,v + p_1\,v_1}{V} = \frac{(25 \times 7) + (18 \times 4\frac{1}{2})}{10} = \frac{256}{10} = 25.6$ lb. per sq. in. Ans.

EXAMPLE.—What must be the volume of a vessel which will hold two gases whose volumes are 7 cubic feet and $4\frac{1}{2}$ cubic feet, and whose

tensions are 25 pounds and 18 pounds per square inch, respectively, in order that the pressure may be 25.6 pounds per square inch, the temperature remaining the same throughout?

SOLUTION.— $V = \frac{p v + p_1 v_1}{P} = \frac{(25 \times 7) + (18 \times 4\frac{1}{2})}{25.6} = 10$ cu. ft. Ans.

GASES COMMON TO MINES.

858. The gases met with in mines are comparatively few in number, but a thorough knowledge of their properties and the manner of their detection is most important. The following are the gases most commonly occurring in mines, considered in the order of their importance, as dangerous to life and health.

Marsh-gas (carbureted hydrogen) (CH_4). Sp. Gr., 0.559.

Marsh-gas (mixed with air)—**Firedamp** (Saturated). Sp. Gr., 0.96.

Carbonic oxide gas (CO)—**White damp.** Sp. Gr., 0.967.

Carbonic acid gas (CO_2)—**Black damp.** Sp. Gr., 1.5291.

Carbonic acid gas....(CO_2)		
Carbonic oxide gas...(CO)		
Nitrous oxide gas...(N_2O)		**After-damp.**
Nitrogen (*free*).......(N)		(Composition very variable.)
Hydrogen (*free*)......(H)		
Watery vapor......(H_2O)		

Sulphureted hydrogen (H_2S)—**Stink damp.** Sp. Gr., 1.1912.

Ethene, or **olefiant gas** (C_2H_4). Sp. Gr., 0.973.

The two latter gases are rarely found in mines, and when present are only in limited volumes.

859. Marsh-gas (CH_4).—This is the most disastrous, in its effects, of any of the gases known to mining.

(*a*) **Occurrence.**—Pure marsh-gas (CH_4), Sp. Gr., 0.559, exists, or *has existed*, as an occluded gas, to a greater

or less extent, in all coal formations, and is a product of the early metamorphism of vegetable matter under the water and superimposed strata, by which all air was excluded. Marsh-gas is often seen rising in bubbles from the bottom of stagnant pools; it is always the product of the decomposition of vegetable matter, away from air, and in the presence of water. When such decomposition of vegetable matter occurs in a *dry* place, away from air, ethene, or olefiant gas (C_2H_4), which is richer in carbon than marsh-gas, is formed.

Marsh-gas transpires from the pores, foliations, and crevices of a freshly exposed face of a gaseous coal-seam. It may also issue from the floor or roof of the seam, as stratigraphical or other conditions may have rendered these adjacent strata more pervious to the gas than the coal-seam itself. It may transpire from the entire face, or may issue in a stronger flow from a crevice or *feeder*. It may even find vent as a *blower* of gas, under great pressure. Natural marsh-gas never occurs in a pure state, but is always mixed with other gases. The mode of occurrence of these gases will be further explained in the study of *Diffusion*, *Occlusion*, and *Transpiration*.

(*b*) **Properties.**—Marsh-gas is a combustible gas, burning with a bluish flame, but it will not support combustion. It is slightly more than one-half as heavy as air of the same temperature and pressure. Upon first transpiring from the face, or issuing from the fissures of a formation, the gas diffuses rapidly in the air, until its limit of diffusion is reached in a confined space, as in the still air of a mine.

Marsh-gas is the lightest of the hydrocarbons, a molecule of marsh-gas consisting of *one* atom of carbon united to *four* atoms of hydrogen. It is an odorless, colorless, and tasteless gas. It does not poison the animal system; a person may breathe with impunity air containing a large percentage of the gas for a considerable time.

One of the most important properties of marsh-gas, to the miner, is that which it possesses of not igniting immediately upon contact with flame. Ignition of the gas takes place

only after the lapse of an appreciable period of time, which, although but a fraction of a second, is sufficient to render the use of many detonating explosives safe in presence of firedamp, the detonation in this class of explosives being instantaneously followed by a period of extreme cold.

(*c*) **Detection of Marsh-Gas.**—On account of the rapid diffusibility of marsh-gas (Table 19), its detection in the mine is practically the detection of firedamp. (See Detection of Firedamp.)

860. Firedamp.—Any explosive mixture of marsh-gas and air is termed *firedamp*.

(*a*) **Occurrence.**—On account of the high diffusive power of marsh-gas, *firedamp* is formed very rapidly wherever marsh-gas issues from the coal or strata. This diffusion takes place upon the outer envelope of the gas in contact with the air. A considerable body of the gas, owing to its lighter weight and warmer temperature, ascends and flows along the roof, collecting in cavities and convenient places for lodgment. The specific gravity of this diffusing gas approaches that of air, and its subsequent diffusion is slow. For this reason, we look for firedamp in the cavities of the roof and the higher working-places of the mine.

(*b*) **Properties.**—The explosive limits of firedamp mixtures can not be closely defined, as such conditions as the purity of the marsh-gas, and the pressure to which the firedamp is subjected, vary the explosive points slightly. However, under ordinary conditions, when 1 part of marsh-gas mixes with $5\frac{1}{2}$ parts of air, the combination is at its lowest explosive limit. As the proportion of air is increased, the explosive violence grows steadily greater till it reaches a maximum, when the mixture is in the proportion of $9\frac{1}{2}$ parts of air to 1 of gas. From this point, as the proportion of air is increased, the explosive violence grows more and more feeble till the mixture consists of 13 parts of air to 1 of gas, when explosion ceases altogether.

The percentage of marsh-gas in an atmosphere of fire-damp, when at its lowest explosive limit, is calculated thus:

Relative volume of gas,	1
Relative volume of air,	5.5
Relative volume of mixture,	6.5

and $\frac{1 \times 100}{6.5} = 15.38$ per cent. of marsh-gas.

In like manner, for the higher explosive limit, we have

Relative volume of gas,	1
Relative volume of air,	13
Relative volume of mixture,	14

and $\frac{1 \times 100}{14} = 7.14$ per cent. of marsh-gas.

The presence in firedamp of $\frac{1}{7}$ of its volume of carbonic acid gas will render it inexplosive.

The effect of an increase of pressure upon the explosive range of gas is to extend it. A mixture of marsh-gas and air that is below or above the explosive limits, is often rendered explosive by an increase of pressure. This may often occur in proximity to a blast when the air of the workings would otherwise be safe.

The effect of suspended coal-dust in the air is to widen the explosive range. This is probably due to the increase of temperature incident to the burning of the gases distilled from the dust.

(*c*) **Detection of Firedamp.**—The detection of this gas in the mine is to be entrusted to the most experienced men only, for it is fraught with danger to all in the mine. Many devices have been invented for the purpose of detecting the presence of gas, as well as to determine at the same time the approximate percentage of the mixture of gas and air. Any machine to be of practical value in this line, must be capable of making the test promptly and safely at the point of danger, and of revealing the presence of $\frac{1}{4}$ per cent. of gas.

We shall refer more particularly to the means at our

disposal for detecting firedamp later, and shall describe the various forms of lamps in common use. We will state here, that, at the present time, no machine or device for testing has given satisfaction equal to the safety-lamp, which is prompt and always at hand. The lamp is elevated cautiously to the place where gas is suspected, care being taken to keep the lamp in an upright position, that its flame may not approach the gauze of the lamp. If gas is present, it will enter the lamp with the air, and will burn when present in large quantities, filling the whole lamp with flame. If the percentage of gas in the air is small, however, say two per cent., its presence is manifested only by a small blue tip to the flame of the lamp, which may be seen more distinctly by screening the eyes from the brighter portion of the flame with the hand.

An experienced and careful observer will detect, with the ordinary lamp, a percentage of the gas as low as 2 per cent. It is, however, often desirable to detect the presence of smaller quantities than this in the air of dusty mines, where the coal-dust is highly inflammable. For this purpose, specially constructed lamps are used. In the use of the lamp for the detection of presence of gas, care must be taken to make no quick movement; especially is this needful in case of flaming in the lamp. The lamp must be immediately removed from the gas, but not so quickly as to blow the flame through the gauze. This requires much self-possession on the part of the observer.

861. White Damp (*CO*).—The "white damp" of the mines is carbonic oxide gas. It is a dangerous gas, because of its harmful effects and its unsuspected presence.

(*a*) **Occurrence.**—Carbonic oxide gas is a product of the incomplete combustion of carbonaceous fuel, the supply of air being limited. Thus, it is produced largely by the slow combustion of coal in the gob, by mine fires, and by the explosion of powder.

(*b*) **Properties.**—This gas is a colorless, odorless, and

tasteless gas. It is somewhat lighter than air at the same temperature and pressure. It burns with a pale, violet flame, like that which may be seen at any time over a freshly fed anthracite fire. It is very poisonous to the system when inhaled, being rapidly absorbed by the blood, and it acts as a narcotic, producing drowsiness or stupor, followed by acute pains in the head, back, and limbs, and afterwards by delirium. If the victim of this gas is not rescued soon, death will inevitably result.

Carbonic oxide gas has the widest explosive range of any gas except hydrogen. When 1 volume of the gas is mixed with about 6.7 volumes of air, the lowest explosive mixture is obtained. From this point it continues to be explosive until the proportion of gas is increased to the extent of 1 volume of gas to every 1.6 volumes of air. It is this property of carbonic oxide gas which makes it such an active agent in the transmission of the flame of a mine explosion from one point in the mine workings to another seemingly isolated point. Under ordinary conditions, however, this gas is not present in sufficient quantity to yield an explosive mixture.

(*c*) **How Detected.**—Carbonic oxide gas may be detected in the mine workings by its effect upon the flame of an ordinary lamp. The flame is much brighter and reaches upwards, and it is thus lengthened out into a more or less slim, quivering taper with a bluish tip, which may be seen more clearly by screening the eyes from the brighter portion of the flame with the hand.

862. Black Damp (CO_2).—The "black damp" of the mines, or, as it is often called, "choke-damp," is carbonic acid gas. It is not as dangerous as either of the preceding gases, because its presence in the mine workings is at once manifested by the dimness of the lamps.

(*a*) **Occurrence.**—This gas is always a product of combustion in the presence of a plentiful supply of air. It is produced by the burning of lamps, breathing of men and

animals, decomposition and decay, and is a later product of all explosions of powder and gas. The principal source, however, is water from the coal and strata which hold it in solution, and from which it escapes as the water evaporates.

(*b*) **Properties.**—This is a colorless and odorless gas, but it possesses a distinctly sharp taste in the mouth when breathed. It is one-half again as heavy as air at the same temperature and pressure, and, therefore, collects near the floor and in the low places in the mine. It is incombustible, and, when present in the air to any considerable extent, extinguishes lamps. It acts as a narcotic, and produces, after a time, headache and nausea, causing death by suffocation.

(*c*) **How Detected.**—The presence of carbonic acid gas is readily detected by the flame of a lamp becoming reduced in size, and, when more gas is present, by its extinguishment; by lime-water, which, when exposed to the gas, becomes milky in appearance; and by damp, blue litmus paper, which becomes red when exposed in an atmosphere containing carbonic acid gas. The flame becomes reduced in size, and, when more gas is present, is extinguished altogether. Being heavier than air, it must be sought for at the floor of the entries and in the low parts of the mine.

863. (*a*) **Traces of Sulphureted Hydrogen Gas** (H_2S).—This gas, though not commonly occurring in troublesome quantities, is yet a very dangerous gas to meet. It is heavier than air, having a specific gravity of 1.1912. It is violently explosive when mixed with air of about seven times its volume. The gas is very poisonous when inhaled. In small quantities in the air, it produces derangement of the system; when inhaled in larger quantities, it rapidly produces unconsciousness and prostration. The smell of the gas affords the best index of its presence, which has given rise to its being termed "***stink damp***" by the miners, for it smells like rotten eggs.

(*b*) **Ethene, or Olefiant Gas** (C_2H_4).—This gas occurs in varying amounts as a constituent of marsh-gas. It is this gas which causes the flame of marsh-gas to burn with some luminosity. It is a product of the dry decomposition of vegetable matter and the distillation of coal.

OTHER PROPERTIES OF GASES.

864. Diffusion.—*All* gases which do not act chemically on each other, especially air and gases *of different densities*, when in proximity, tend to *diffuse* into each other; that is to say, their molecules pass freely among each other, and tend to form a complete intermixing of the two gases. This property is called *diffusion*, and is caused by the lack of equilibrium between the molecular vibrations of the two masses; so that the molecules of the two masses tend to thoroughly intermingle. (See Art. **856.**)

865. Rate of Diffusion.—The diffusion of gases takes place much more rapidly in a moving current than in still air. The relative rates or velocities of the diffusion of the gases into each other are in the inverse ratio of the square roots of their densities. For example, taking the density of air as 1, then the density of hydrogen gas, by Table 19, is .0693, and the square root of .0693 by the table is .2632; therefore, the relative velocity of the diffusion of hydrogen gas into air will be $\frac{1}{\sqrt{.0693}} = \frac{1}{.2632} = 3.7987$. This corresponds with the results given in the third column of the table; and the use of the table may be understood in this way. In all cases, divide 1 by the square root given in the second column of the table, and the quotient will be the relative velocity of the diffusion of the gas in question into air. For example, the square root of the density of marsh-gas is given in the second column as .7477; then, $\frac{1}{.7477} = 1.3375$ = the relative velocity of the diffusion of marsh-gas into air. The annexed table of densities shows the

comparative rates of diffusion of the various gases and air into a vacuum:

TABLE 19.

Gas.	Density, or Specific Gravity.	Square Root of Density.	$\frac{1}{\sqrt{\text{Density}}}$.	Velocity of Diffusion. Air = 1.
Hydrogen	0.0693	0.2632	3.7987	3.830
Marsh-gas	0.5590	0.7477	1.3375	1.344
Carbonic oxide	0.9670	0.9834	1.0169	1.015
Nitrogen	0.9713	0.9855	1.0147	1.014
Oxygen.............	1.1057	1.0515	0.9510	0.949
Sulphureted hydrogen..............	1.1912	1.0914	0.9163	0.950
Carbonic acid......	1.5291	1.2366	0.8087	0.812

The values given in the last column of this table were obtained by experimenting with the gases, and agree quite closely with the calculated values given in the preceding column. From the last column we see that 1,344 volumes of marsh-gas will diffuse in the same time as 1,000 volumes of air or 812 volumes of carbonic acid gas.

866. Occlusion of Gases.—A gas is *occluded* (hidden) when it exists in the pores of a solid mass. A familiar example of the occlusion of gases is found in the coal-seams, where gases often exist in large quantities and are a source of danger in mining.

The conditions which have held these gases in the coal and adjoining strata, till set free by the penetration of mine workings, are largely a close coal and an impervious roof and floor. The kind and amount of gases occluded in different coal-seams, and even in different parts of the same seams, vary much, and alter, to a large extent, the character of the coal enclosing them.

The gases most commonly occluded in coal-seams are marsh-gas, nitrogen, carbonic acid gas and traces of oxygen, carbonic oxide, ethene, and some other hydrocarbons.

The relative percentages of these gases vary largely, even in freshly mined coals.

867. Pressure of Occluded Gases.—The pressure of occluded gases has been shown, by a number of experiments in England, France, and Belgium, to reach as high as 10 and 16 atmospheres; and, in exceptional cases, 32 atmospheres has been the recorded pressure. Whatever degree of exactness these experiments may have, they serve to show, at least, the enormous pressures under which occluded gases may be projected from a newly exposed face. In some instances, this flow of natural gas from certain veins has furnished fuel for extensive steam plants. In general, the tapping of a gaseous seam relieves the pressure, after a limited time, by the escape of the gas.

The pressure of occluded gases is often manifested in a newly exposed face of coal by a sharp cracking and hissing sound, throwing the splintered coal with considerable violence into the face of the miner.

868. Transpiration of Gases from Coal.—When a coal-seam containing occluded gases is being worked, the pressure on the gas drives it outwards from the coal, and often from the roof and floor of the seam. The regular emission of gas from a solid mass in which it was contained is called *transpiration.*

869. Feeders and Blowers.—Wherever a cavity, crevice, or fissure exists in proximity to or in connection with a gaseous seam, it becomes charged with the occluded gases of the seam, under the same pressure. A dangerous reservoir of gas is thus formed, which may at any moment be pierced or tapped by the pick or drill of the miner and discharge its contents into the mine workings. Such *cavities*, *crevices*, or *fissures* charged with gas are termed "*feeders*," and, when tapped, the stream of gas issuing from them is called a "*blower.*" According to the size of the internal reservoir of gas, such a blower may continue to discharge its gas, with practically no abatement, for a long time.

870. Outbursts.—In the working of the seams of some localities, the presence of occluded gases is frequently manifested by a violent outburst at the working face. These outbursts often take place without warning, and produce an effect similar to that of an explosion, throwing down the coal in large quantities.

The cause is due to a *feeder* finding access to a more or less vertical crevice or cleat behind the working face of the coal-seam. Its pressure thus becomes distributed over a considerable area of coal, and exerts a powerful localized force. This is due to a pressure on a large area being made to act on a small one with multiplied force.

Fig. 113 represents a dangerous pocket of gas lying beneath an impervious stratum of close-grained rock, which has prevented its escape. The gas is under enormous pressure, incident to the subsidence and contraction of the strata. The cleats or vertical fissures shown in the coal-seam are "face cleats," the entry or gangway being driven "end on." The pressure of the gas causes the foliated shale to rest heavily on the timbers, and later a fissure occurs in this strata, which opens a communication for the gas with the face cleats of the coal. The pressure of the gas may thereby be distributed over a large area of the rib, with the result just described.

Fig. 113.

There are well-authenticated, although seemingly incredible, cases upon record where headings and chutes have been completely blocked by a compacted mass of from 15 to 20 tons of fine coal, thus thrown from the face without the slightest warning. In other instances, the outburst may be accompanied by a subterranean pounding, or "bumping,"

as the miners term it, or by a sudden report, similar to that of a blast. This pounding or "bumping" sometimes continues at intervals for two or three days prior to the outburst. By far the larger number of violent outbursts are of marsh-gas; although instances are recorded of very violent outbursts of carbonic acid gas.

871. Calculation of the Initial Force of an Explosion.—The *force* of an explosion depends upon the expansive power of the gases resulting from the explosion. The expansive power of these gases depends upon their relative volumes *before* and *after* the explosive reaction has taken place. The *initial force* of an explosion is the force developed at the moment of ignition. To calculate the initial force of an explosion, we must first determine the *relative* or *atomic volume* (Art. **841**) of the resulting gases. Thus, in the *complete* explosion of marsh-gas (CH_4), the reaction which takes place is expressed by the following equation:

$$CH_4 + 4\,O + 16\,N = CO_2 + 2\,H_2\,O + 16\,N,$$

or 1 atom of carbon + 4 of hydrogen + 4 of oxygen + 16 of nitrogen = 25 atoms; and 1 atom of carbon + 2 of oxygen + 4 of hydrogen + 2 of oxygen + 16 of nitrogen = 25 atoms.

We notice in this *complete* explosion, the maximum explosive energy must be developed, because *all* of the carbon of the marsh-gas is converted immediately into carbonic acid gas, and *all* of the hydrogen into watery vapor, both of which are dead or inert products, having given out their energy. We shall also see later that this occurs when the marsh-gas forms 9.38 per cent. of the firedamp.

By observing the above equation, we see that for each molecule of CH_4 there is produced *one* molecule of CO_2 and *two* molecules of $H_2\,O$. *Four* atoms of O are consumed in the reaction, which are derived from the air and represent a volume of air containing approximately 4 atoms of O and 16 atoms of N. (More accurately, the 4 atoms of O represent 20.7 per cent. of the entire air used; see Art. **845**.)

Now, determining the atomic volume of the gases before and after the reaction, we find the volume of the firedamp (marsh-gas and air) is the same as the volume of the carbonic acid gas, watery vapor, and nitrogen produced; thus, the molecule of marsh-gas occupies the same space as the molecule of carbonic acid gas; the *two* molecules of watery vapor occupy the same space as the *four* atoms of oxygen. (See Second Law of Volume, Art. **841.**) The nitrogen is unchanged by the reaction.

Before Explosion.				After Explosion.			
Gas.	Symbol.		Atomic or Relative Volumes.	Gas.	Symbol.		Atomic or Relative Volumes.
Marsh-gas.	CH_4	1 molecule.	2	Carb. acid gas.	CO_2	1 molecule.	2
Air.	O	4 atoms.	4	Watery vapor.	$2\,H_2O$	2 molecules.	4
	N	Free nitrogen.	15.32	Nitrogen.	N	Free nitrogen.	15.32
		Total volume,	21.32			Total volume,	21.32

By observing the above table, we see that the column of relative volumes shows 2 volumes of marsh-gas and $4 + 15.32 = 19.32$ volumes of air, which is in the ratio of 1 volume of marsh-gas to 9.66 volumes of air, the firedamp being at its maximum explosive point. We observe, also, by the table, that 4 volumes of oxygen are consumed in the complete explosion of 2 volumes of marsh-gas. We have previously learned (Art. **845**) that oxygen forms 20.7 per cent. of the volume of the air; the remaining 79.3 per cent. being nitrogen in a free state. Hence, to find the relative volume of air per 2 volumes of gas, we write the following proportion : $20.7 : 4 :: 100 : x = 19.32$ volumes of air, or, for 1 volume of gas, we have $20.7 : 2 :: 100 : x =$

9.66 volumes of air. The entire relative volume of firedamp concerned in the reaction is, then, the sum of the relative volumes (21.32 volumes), as by the table.

The percentage of pure marsh-gas in this mixture, or body of firedamp, is found thus, $\frac{2 \times 100}{21.32} = 9.38$ per cent. of marsh-gas. There being no change in the atomic volume of these gases before and after explosion, the expansive power produced by the combustion of the mixture can be found from the increase in temperature from 60° F. (normal) to 1,200° F. (temperature of ignition for marsh-gas). Thus, the total pressure of a confined gas is always proportional to its absolute temperature. The absolute temperature is the temperature above absolute zero, which is 459° below the Fahrenheit zero. Hence, to transform Fahrenheit temperature to absolute temperature we add 459 degrees. Then, knowing the pressure of the atmosphere to be 14.7 at the normal temperature 60° F., we write the simple proportion

$$459 + 60 : 459 + 1{,}200 :: 14.7 : x,$$
$$\text{or, } 519 : 1{,}659 :: 14.7 : x = 47.0 \text{ pounds, nearly.}$$

Therefore, the absolute pressure (or the pressure above vacuum), after the explosion, is practically 47 pounds, and the ruptive pressure is $47.0 - 14.7$ (or the atmospheric pressure) $= 32.3$ pounds per square inch.

872. Calculation of the Weight of a Gas.—*The weight of any gas, at a given pressure and temperature, is equal to the weight of an equal volume of air, at the same pressure and temperature, multiplied by the specific gravity of the gas.*

Let W = weight in pounds;
V = volume in cubic feet;
B = barometric pressure in inches;
D = specific gravity of the gas—found in Table 19;
T = absolute temperature.

Then, $$W = \frac{1.3253\,V\,B\,D}{T}. \qquad \textbf{(21.)}$$

EXAMPLE.—What is the weight of 100 cubic feet of carbonic acid gas at a pressure of 31 inches of mercury and a temperature of 32° F.?

SOLUTION.— $W = \dfrac{1.3253 \times 100 \times 1.5291 \times 31}{459 + 32} = 12.7947$ lb. Ans.

NOTE.—The constant 1.3253 is the weight, in pounds, of 1 cubic foot of air at 1° absolute temperature (Fahr.) and 1 inch barometric pressure.

EXAMPLES FOR PRACTICE.

1. Find the weight of 200 cubic feet of marsh-gas (CH_4), at a temperature of 70° and a pressure of 30 inches of mercury. Ans. 8.4028 lb.

2. Referring to example 1, what is the weight of the hydrogen gas, in this amount of marsh-gas? Ans. 2.1007 lb.

3. In case of an explosion of firedamp in which 8.4028 pounds (example 1) of marsh-gas were concerned, all of its hydrogen combining with the oxygen of the air to form water (H_2O), (*a*) what would be the weight of watery vapor resulting from the explosion? (*b*) What would be the weight of oxygen consumed in this part of the reaction? (See Art. **838.**)

Ans. { (*a*) 18.9063 lb. (*b*) 16.8056 lb.

4. Referring to example 3, if all of the carbon of the 8.4028 pounds (example 1) of marsh-gas, combined with the oxygen of the air to form carbonic acid gas (CO_2), (*a*) what weight of carbonic acid gas would result from the explosion? (*b*) What would be the weight of oxygen consumed in *this* part of the reaction? (See Art. **838.**)

Ans. { (*a*) 23.1077 lb. (*b*) 16.8056 lb.

5. Referring, now, to examples 3 and 4, (*a*) what is the total weight of oxygen consumed in the reaction? (*b*) Determine the total weight of air consumed in the reaction, incident to the explosion. (See Art. **845.**)

Ans. { (*a*) 33.6112 lb. (*b*) 146.1356 lb.

6. What volume of dry air is required to completely explode 200 cubic feet of marsh-gas? (See Arts. **841** and **845.**)

Ans. 1,932.3 cu. ft.

7. Referring to example 6, if this volume of air (1,932.3 cubic feet) is consumed in the complete combustion of 200 cubic feet of marsh-gas, (*a*) what per cent. of the mixture (firedamp) does the marsh-gas form? (*b*) What volume of firedamp was exploded?

Ans. { (*a*) 9.38 per cent. (*b*) 2,132.3 cu. ft.

COMBUSTION.

873. Combustion, in its broadest sense, refers to chemical union, attended with *heat*, sometimes with *light* and *flame*. Combustion always results in a complete transformation of the body acted upon, and of the gas which *supports the combustion*, forming other gases.

874. Oxidation.—Oxygen gas is the great supporter of combustion; and the process of combustion is then called *oxidation*. This gas has a strong affinity for carbon and hydrogen; and, thus, we have formed two of the most commonly occurring compounds, *water* and *carbonic acid gas*. The *first* of these is as truly an essential to animal life as the *second* is an inevitable result of the same.

There are numerous other illustrations of true combustion, however, than those in which *oxygen* plays a part. For example, the burning of a lighted taper in an atmosphere of *chlorine* gas; or, the explosion of equal quantities of *hydrogen* and *chlorine* gases.

875. Temperature of Combustion.—Combustion may take place at any temperature; that is to say, the oxidation is often carried on slowly and at a low temperature, and the body just as truly destroyed or consumed as when the action is stronger and the temperature high enough to produce flame.

The process is then spoken of as **slow combustion,** because the action is slower, or less energetic than in **active combustion,** when flame is produced. The consuming of the animal tissues of the body is an example of *slow combustion*. The disintegration of fine coal, in the gob heaps and goaves of the mine, is followed, in time, by a slow oxidation of the coal and the formation of carbonic oxide and carbonic acid gases. This slow oxidation is as truly a form of combustion as when the coal is burned at a higher temperature and flame results.

We conclude, then, that a high *temperature* is not an essential to slow combustion. The chemical activity of any combustion will determine its initial temperature; on the

other hand, the various products of combustion have varying heat capacities (specific heats), and thereby absorb varying amounts of the initial temperature, the remaining difference being the *sensible* heat of the combustion.

876. Spontaneous combustion is a term applied to the sudden bursting forth of flame, or active combustion, in a body, caused by the internal generation of heat in the body itself. Spontaneous combustion is the result of slow combustion, or chemical action, the developed heat gradually increasing till ignition takes place. The production of carbonic oxide gas (CO) within the confines of the body, where the supply of air is limited, greatly assists ignition.

EXPLOSIVES AND EXPLOSIONS.

877. Explosives.—This term refers to any chemical compound or mechanical mixture that is capable, under certain conditions of heat or shock, of exerting a powerful ruptive pressure. The common form of explosives in use is an intimate, mechanical mixture of chemical compounds, such as will readily give rise to dissociation of their respective atoms, and the rapid formation of gases, under the proper conditions. These gases, from the temperature incident to the explosion, possess an enormous expansive force, resulting in a ruptive pressure of many tons upon the square inch.

On account of the great importance of explosives in mining, and on account of their being the direct cause of a large number of mine accidents, affecting many who are in no wise to blame for their occurrence, a careful study of their nature and use is needful. We will consider, in order, the conditions incident to the explosion of a charge in a drill-hole, for the purposes of blasting.

The chief factor which determines the strength of an explosive is the rapidity of its combustion. There are two modes of propagation of the combustion of explosives, giving rise to two general classes; in the first class the propagation being slow, while in the second it is extremely rapid.

(*a*) *Explosives that deflagrate.*
(*b*) *Explosives that detonate.*

878. Deflagration is a form of combustion dependent upon the thermal conductivity of the mass. The combustion is propagated from particle to particle of the mass, much as heat travels from one end of an iron rod to the other. All the black powders furnish examples of *deflagration*. The ignition of such a powder at one point is transmitted throughout the mass with a speed dependent upon the combustibility of the powder and its thermal conductivity. Each atom burns independently, and exerts no further influence upon the surrounding atoms, except as the heat of its burning is communicated to them.

879. Detonation.—This form of combustion, unlike deflagration, is transmitted, with almost lightning rapidity, to every particle of the mass. The detonation of a single particle seems to exert a wave-like compression throughout the mass that causes a like detonation of the entire body. Its speed of propagation is estimated at 16,400 feet per second; so that any explosion by *detonation* is, practically, an instantaneous development of the entire expansive energy of the mass. Nitroglycerine is an example of such an explosive.

880. Action of Explosives.—The theory of the action of explosives is, in outline, as follows:

Chemical action, incident to the ignition of a charge, assisted by *heat* and *pressure*, transforms the solid explosive compound or mixture into gaseous products, developing in such transformation or *combustion* a definite number of heat units.

881. Chemical Reaction.—When a charge of powder is exploded in a drill-hole, the combustion which takes place is supported by the oxygen of the niter in the powder. This salt is a powerful oxidizer, and gives up its oxygen to the sulphur and carbon. A large number of gases are formed, chief among which are nitrogen, carbonic acid gas, and carbonic oxide gas. It is impossible to give any accurate analysis of the gases resulting from any one explosion. The gaseous products vary according to the pressure

under which ignition takes place, and will vary, therefore, in each individual charge. Any chemical equation, therefore, expressing the reaction which takes place must be only approximate.

882. Black Powders.—The better grades of the black powders are formed by the intimate mixture of 2 molecules of niter, 1 of sulphur, and 3 of fine charcoal, making the following proportions by weight:

Niter (saltpeter), (potassium nitrate)	(KNO_3)	= 74.83%
Sulphur............................	(S)	= 11.84%
Carbon (fine charcoal)..............	(C)	= 13.33%
		100.00

In practice, however, the proportions are usually taken as follows:

Niter..................	75 parts.
Sulphur	10 parts.
Carbon	15 parts.
	100 parts.

883. Blasting Powder.—It is common practice, in the manufacture of *blasting* powders, to increase the amount of carbon or charcoal, while the amount of the niter is decreased. In blasting powders, the following proportions are more commonly used, although this practice varies in different localities:

Niter................	66 parts.
Sulphur..............	10.5 parts.
Carbon	23.5 parts.
	100 parts.

Cheaper grades of blasting powders are often made by substituting sodium nitrate $(Na\,NO_3)$ for the potassium nitrate, either in part or wholly; but such substitution produces a very inferior powder. The sodium salt absorbs moisture when exposed to even the slightest dampness, and thereby causes such powders to lose much of their strength.

884. Size of Grain.—The size of the grain is an important factor in determining the rapidity with which the powder acts. The finer the grain, the quicker the action; and *vice versa*, the coarser the grain, the slower the action. A little study will show the need of a careful application of this principle, on the part of the miner. For example, *gunpowder* is a fine-grained powder; what is desired is the *rapid* movement of a *small* mass; hence, its action must be very rapid, and the stock of the gun must be strong enough to resist the inertia of the bullet.

On the other hand, in all kinds of blasting, a *slower movement* of a *larger mass* is desired; hence, we employ a slower powder, one whose whole expansive energy will not be developed in a single flash. We must consider, also, that the blasting of different materials requires a different action in the powder, according to the character of the material blasted. Thus, the blasting of rock requires a quicker powder than the blasting of coal, while a soft, laminated shale will yield more completely to a very slow powder. The need for this adaptation of size is obvious and reasonable.

Fig. 114 shows, approximately, the four sizes of black powder in most common use in coal-mining. These sizes

FIG. 114.

are adapted to different grades of work and a varying hardness in the coal. The smaller sizes are adapted to a hard, brittle coal, while the larger sizes are, on the other hand, adapted to a softer and tougher coal. The smaller sizes are likewise adapted to *narrow work* (entry work) and to *shooting on the solid*, while the coarser grades yield better results in *breast* and *pillar* work, where the resisting forces are not so great. The nature of the coal, the class of work, and the judgment of the miner must determine the size of

powder best adapted for his use. Many miners, however, use very poor judgment in this respect, and reap a reward in the decrease of their net earnings.

885. "Blown-out" Shot.—This term is applied to any blast whose energy is expended upon the air, instead of being converted into mechanical work. The intensity of the projected flame is augmented by the high temperature and pressure resulting from the unyielding character of the walls enclosing the charge.

886. "Windy" Shot.—This term refers to a blast whose energy is, in part or wholly, expended upon the air. It differs from a *blown-out* shot only in the absence of the high temperature and pressure of the projected flame.

887. Causes.—The causes giving rise to the above are numerous, and may be summarized as follows:

(*a*) The shot may be too deeply laid.

1. The angle of a *gripped* shot may be too large; that is, the hole may be drilled at an angle so great that the charge will lie too deep.

2. The depth of a hole may locate the charge too much upon the *solid* (back of the *cutting* or *mining*).

3. The projecting bottoms and tops of the seam may arch the resistance in such a manner as not to allow the charge an opportunity to do its work.

(*b*) 1. The tamping (stemming) may be insufficient for the charge exploded or the sectional area of the hole.

2. The tamping may be of such an inflammable and gaseous nature as to become a dangerous factor in lengthening out the flame of the blast by the gases distilled from it under the flame of the blast.

(*c*) 1. The *solid*, in the region of the charge, may be creviced or fissured naturally, or by a former blast.

2. The coal may "seam out."

(*d*) 1. Too strong (fine-grained) a powder may be employed, which results in blowing the tamping and giving

vent to the flame and gases of the blast before the inertia of the mass has been overcome.

2. Too coarse a powder and too heavy a charge of it may result in a considerable amount of partly burned and burning powder being thrown out upon the air, to expend its energy in expansion instead of in mechanical work. A like result will always be produced by an *excessive* charge of any size powder.

3. A mixture of different grades of powder will nearly always result in a considerable portion of the charge being thrown upon the air, partly burned or burning. The mixing of a small amount of gunpowder with blasting powder, for the purpose of "making it stronger," is a pernicious act, and would justify the discharge of the man found guilty of so doing.

(*e*) A drill-hole of too large a diameter, as compared with the amount of the charge, will result, in the majority of cases, in the projection of the charge, because the large sectional area of the hole brings an undue pressure upon the tamping.

(*f*) 1. A succession of two or more blasts, fired in a limited working place, may produce an effect similar in every respect to that produced by a *windy shot.* It is caused by the firing of the carbonic oxide gas and the suspended dust of the first shot, by the flame of the second.

2. A like result obtains very often when a heavy blast is fired in too close proximity to accumulations of dust.

In general, if the hole is "gripped" too strongly, or the charge itself located too deeply upon the solid, a "blown-out" shot will result from the unyielding nature of the walls, and a flame of great intensity will be projected from the bore of the hole when the tamping or stemming has yielded.

If the charge is too heavy for the work to be accomplished, a "*blown-out,*" but more properly called, a "*windy*" *shot,* will result. The temperature of the flame will be normal in this case, but the danger arises from the projection and explosion of a considerable amount of the charge upon

the air after rupture has taken place. The energy of a portion of the charge is thus bestowed upon the air instead of being converted into mechanical work, by breaking down the coal.

888. Flameless Explosives.—From our previous study, we readily perceive the dangers incident to blasting in mine workings. So numerous are the conditions which render the use of explosives in a mine dangerous, that it has often been a matter of serious consideration whether the use of any form of explosives should be tolerated in mines known to be gaseous. It is recognized that the *flame* incident to the explosion is the dangerous factor, and many attempts have been made to so alter the composition of the explosive as to yield gaseous products which were not inflammable. This result has only been realized in part. Nevertheless, explosives have been produced in the combustion of which a very limited flame results; and the use of such explosives renders mining more safe. These are mostly formed by a mixture of nitrated compounds (ammonia nitrate and nitro-benzine, or nitro-naphthalene). For the most part, they are detonators, and are exploded by a fulminating cap.

DETONATING EXPLOSIVES.

889. The detonating explosives are divided into three general classes, viz.:

(*a*) Such as have *glycerine* for a base, as *nitroglycerine*, *dynamite*, *carbonite*, *stonite*, and *ardcerite*.

(*b*) Such as are formed from *cotton*, as *guncotton*, *tonite*, and *potentite*.

Gelignite and gelatine-dynamite (blasting gelatine) are formed by mixing nitroglycerine with guncotton, in varying proportions.

(*c*) Such as have ammonia nitrate for a base (called the Sprengel class, after their inventor), as *Roburite*, *Securite*, *Ammonite*, *Oxonite* (*Rack-a-rock*), *Panclastite*, *Bellite*, and *Hellhoffite*.

890. Nitroglycerine is a heavy, oily liquid, formed by the action of a mixture of strong nitric and sulphuric acids upon glycerine. It is a chemical compound, and as such differs from most other explosives. The dissociation of atoms takes place instantaneously throughout its mass, and thus affords one of the most powerful explosives known.

Its specific gravity is 1.6. It freezes at 40° F. Heated to 360° F., it either burns or explodes. One volume of nitroglycerine exploded yields 1,298 volumes of gas. Nobel places the temperature of the explosion at 3,270° F., and states that the *expanded* gases of the explosion will occupy 10,384 times the original volume, which will develop a ruptive pressure of 76.322 tons per square inch, under ordinary conditions.

Nitroglycerine, when frozen, will not explode by any ordinary cause; but an elevation of temperature makes its handling dangerous. It is readily exploded by a smart blow, when spread upon a flat surface; but a bottle of the liquid may be smashed to pieces, at times, without causing an explosion. When nitroglycerine has become sour and impure, spontaneous decomposition is developed, forming gas and oxalic acid, which often results in a disastrous explosion, especially when the liquid is contained in a tightly-stoppered vessel.

Nitroglycerine is rendered more safe for blasting purposes and for transportation, by its being employed in the form of *dynamite.*

891. Dynamite.—This explosive is nitroglycerine, absorbed by any porous substance. There are different grades of dynamite, differing by the varying amount of nitroglycerine absorbed. They are rated as follows, the percentages varying according to the different brands :

Grade No. 1, from 50 to 70 per cent. nitroglycerine.
Grade No. 2, from 33 to 50 per cent. nitroglycerine.
Grade No. 3, from 27 to 30 per cent. nitroglycerine.
Grade No. 4, from 20 to 25 per cent. nitroglycerine.

The principal brands in use are "Hercules," "Atlas," and

"Ætna." The dynamite cartridges consist of strong paper shells, previously dipped in melted paraffine, and filled with the explosive. They are usually 8 inches long and of the following diameters and weights:

Diameter $\frac{7}{8}$ inch	Weight about 4 ounces.
Diameter 1 inch	Weight about 5 ounces.
Diameter $1\frac{1}{4}$ inches......	Weight about 8 ounces.
Diameter $1\frac{1}{2}$ inches......	Weight about 12 ounces.
Diameter $1\frac{3}{4}$ inches......	Weight about 15 ounces.
Diameter 2 inches......	Weight about $1\frac{1}{4}$ pounds.
Diameter 3 inches......	Weight about 3 pounds.
Diameter 4 inches......	Weight about 5 pounds.

The weight of any dynamite cartridge may be calculated by means of the following simple rule:

Rule.—*Multiply the square of the diameter of the cartridge by its length, all in inches, and take $\frac{5}{8}$ of the product; the result will be the weight of the cartridge in ounces.*

Let W = weight of cartridge (ounces);
d = diameter of cartridge (inches);
l = length of cartridge (inches).

Then, $$W = \frac{5}{8} l d^2. \qquad \textbf{(22.)}$$

An average No. 2 grade of this explosive will yield an initial ruptive pressure of 24 tons per square inch.

Safe methods of using dynamite are explained further on, in the section on Shafts, Slopes, and Drifts.

Other forms of dynamite have been invented and brought forward from time to time. These mostly consist of nitroglycerine, in smaller quantities, absorbed in various waste products, as cork shavings, sawdust, etc. In the original dynamite, the absorbent was an infusorial earth found in northern Germany, which absorbed three times its own weight of nitroglycerine. The forms of dynamite referred to above are known as *carbonite*, *stonite*, and *ardeerite*.

892. Guncotton (nitro-cotton) is a product similar in all respects to nitroglycerine, being formed by the action of

a mixture of strong nitric and sulphuric acids upon ordinary cotton, or cellulose, wood-pulp, paper, or rags. In appearance, guncotton resembles ordinary cotton; 100 parts, by weight, of cotton should form 183 parts of guncotton; but, on account of more or less incomplete action, and a solution of a portion of the guncotton, before the whole mass has been converted, in practice 100 parts of cotton yield only from 160 to 178 parts of guncotton.

	Exploded in Free Air.	Detonated Under Pressure.
Carbonic oxide gas	30 parts	40
Carbonic acid gas	20 parts	25
Marsh-gas	10 parts	Trace
Nitrogen dioxide	9 parts	None
Nitrogen	8 parts	15
Hydrogen	None	20
Aqueous vapor	23 parts	None

Guncotton is exploded by percussion. In some cases, it has been known to explode with violence when heated to 110° F., although other instances are recorded where the temperature has been raised to 200° F. without an explosion taking place. It has been known to be exploded by the heat of the sun's rays. It is liable to decompose, which often results in spontaneous combustion. Exploded, it yields a gaseous product consisting of 100 parts, as shown in the above table.

As will be readily seen, from its gaseous products, it is not adapted for use in mine workings. Its explosive force, as compared with an equal weight of gunpowder, is as 4.5 to 1.

893. *Tonite* and *potentite* are forms of guncotton to which nitrates of potassium, or barium have been added.

894. *Gelatine-dynamite*, or blasting-gelatine, and also gelignite, are mixtures of nitroglycerine and guncotton, on

the supposition that a more perfect combustion is thereby obtained. A honey-colored, gelatinous mass is obtained, which does not freeze as readily as nitroglycerine or dynamite, and withstands the action of water better. It is more liable to explosion from a sudden blow than is dynamite. Its gaseous products prevent its general use in mining.

895. Sprengel Explosives.—What may very properly be called the *Sprengel explosives*, after their inventor, are the highly nitrated compounds formed by varying mixtures of nitrate of ammonium, $(NH_4)NO_3$, which contains 60 per cent. of its weight of oxygen, with other nitrated compounds, as nitro-naphthalene, nitro-benzol, etc. The explosives belonging to this class are of recent invention and are not well known; but, on account of the property which they all possess, to a greater or less extent, of suppressing the *flame* of their explosion, they will eventually find an important application in mining (Art. **888**). The most important and best known of these are *roburite*, *securite*, *ammonite*, *oxonite*, called also *rack-a-rock*, and *bellite*. The first three of these are alluded to as exceedingly safe and powerful explosives, by G. W. Wilkinson and other competent authorities.

896. Comparison of Explosives.—The value of an explosive lies in its being instantly convertible into gaseous products, having a high temperature and being incombustible. The explosive that embodies these qualifications to the highest degree is the strongest. However, except in very gaseous mines, high explosives are not used in coal-mining, because they shatter the coal and make too much small coal and slack. The less powerful and slower black powder is used, as it breaks down the coal in larger lumps.

(*a*) It is necessary, in order to secure the greatest rending force in an explosive, that its transformation into the gaseous state should be *instantaneous* and *complete*.

(*b*) The higher the temperature developed in the explosion, the greater will be the expansive force of the gaseous products.

(*c*) The more incombustible the gaseous products, the less flame will be produced by the explosion, and the more security will attend its use in gaseous mines.

The following table will be of use to the mining student, in making comparisons between some of the more common explosives in use in mines.

TABLE 20.

Explosive.	Temperature of the Explosion (F.).	Products of Explosion.		Ruptive Pressure (Pounds per Sq. In.)
		Combustible.	Incombustible.	
Blasting powder..	2,000° to 3,600°	42 per cent.	58 per cent.	12,400 to 20,500
Nitroglycerine ...	5,740°	0 per cent.	100 per cent.	152,640
Dynamite	5,280°	0 per cent.	100 per cent.	48,000
Blasting-gelatine..	5,830°	46 per cent.	54 per cent.	....
Guncotton	4,800°	61 per cent.	39 per cent.	90,000 to 100,000
Tonite	4,800°	8 per cent.	92 per cent.	....
Roburite	3,800°	0 per cent.	100 per cent.	....
Ammonite	......	0 per cent.	100 per cent	....
Securite..........	......	0 per cent.	100 per cent.	....
Carbonite	......	41 per cent.	59 per cent.	....

Table 21 gives the temperature of combustion of some of the more important gases relative to mining chemistry.

TABLE 21.

Gases.	Temperature of Combustion (F.).
Marsh-gas..................	1,220°
Ordinary illuminating gas ..	1,198°
Carbonic oxide gas.........	1,184°
Hydrogen.................	1,148°

897. Character of Mine Explosions.—Many conditions influence and determine the character of an explosion

of mine gases. The term *explosion*, in its present application. is broadened to include any type of rapid combustion of mine gases in the air-passages or workings, from a quiet burning, sweeping the roof of the passage and advancing at a moderate velocity, to a wild hurricane of fire, dust, and débris, propelled at an inconceivable speed by the expansive energies caused by the ignition of the gases in the air. The conditions that thus determine the character of an explosion of mine gases are, briefly, as follows:

(*a*) The **proportionate mixing of the gases** and their affinities for each other when excited by heat produces the violence of their dissociation and recombination in other forms as compounds.

For example, the explosive mixture may be air charged with marsh-gas, in such proportions as to develop its maximum violence; or, on the other hand, a large proportion of carbonic oxide gas may be produced as a result of a local explosion of marsh-gas, and this gaseous mixture may burn quietly along the roof of a passageway without exploding. Again, these conditions may suddenly change, and the slow burning at any moment develop explosive violence by contact with another body of gas.

(*b*) The **oxygen of the air** being the ever-ready means to dissociation, the abundance of its supply in the air of the workings determines largely the chemical activities.

(*c*) **Coal-dust,** suspended in the air of mine workings, acted upon by the flame of an explosion, distils **carbonic oxide gas.** This gas has the effect of lengthening the flame, which feeds upon it, and thereby propagates an otherwise local explosion.

(*d*) The physical surroundings of an explosion of mine gases, such as the size of the working places, and all the conditions which hinder the free expansion of the gases, affect the **pressure** and **temperature** of the explosion. These are important factors in determining the products of the explosion and the extent of the flame.

898. Causes of Ignition.—These are many. The ignition of an explosive mixture of gases requires some cause that will raise its temperature to the point of ignition. In the case of firedamp, however, this temperature must be maintained for a certain fraction of a second, or the gas fails to ignite. This is a very important point, as upon it depends the security of detonating explosives.

For example, the initial temperature of the explosion of dynamite is 5,280° F. (Table 20); but so rapid is the propagation of the combustion in the dynamite, that this temperature is only maintained for a time not exceeding $\frac{1}{1000}$ of a second, when its heat is converted into mechanical work, the temperature falling simultaneously with the expansion far below the point of ignition for firedamp (1,220° F.), and thus failing to ignite this dangerous mixture. The interval of time necessary for the ignition of firedamp is probably due to the absorption of heat by the watery vapor formed by the dissociation, and which must be converted into steam at a high temperature before ignition of the gaseous products can take place.

In the case of the ignition of a body of gas (firedamp), the cause is usually the flame of a naked lamp, or a defective safety-lamp, or the flame incident to blasting.

In the case of an explosion in a *non-gaseous* mine, the gases which enter into the explosion are derived from the distillation of the coal-dust suspended in the air, and, in a measure, also from the fine coal pulverized by the crushing force of the blast. In this latter case, the cause of ignition is plainly the projected flame of a "*blown-out*" shot, which has a volume and intensity sufficient for the conversion of a large body of suspended dust into gas.

899. Temperature of an Explosion.—In any explosion whatever, whether it be a body of gas in the mine workings, or a charge of powder, or other explosive, in a drill-hole, the primary or initial temperature of the reaction is determined from the *heat units*, stored in the original constituents of the explosive mixture, and the *specific heats*

or *heat capacities* of the resulting products of the explosion. This *temperature of ignition* may be calculated from the principles of thermal chemistry, and is always a *fixed* temperature, as far as the explosive is concerned.

The *temperature of the explosion*, on the other hand, is determined or influenced by other causes, and is always, to a greater or less extent, lowered by external causes; as, for example, (*a*) loss of heat, by conduction, before the full development of the explosion; (*b*) loss of heat, by absorption due to expansion, before the full development of the explosion.

It will be readily seen that these losses are larger, the slower the progress of the combustion. Thus, in the case of a deflagrating charge, as of *black powder*, whose temperature of *ignition* is 3,632° F., the temperature of *explosion*, depending upon the strength of the resisting walls, is lowered to a practical 2,000° F. In the case of the quiet burning of a body of firedamp, diluted below the explosive point, or the burning of a trail of carbonic oxide gas, left in a passageway at times by the quick advance of an explosion, and fed later by fresh air from rooms or chambers, the *effective* temperature of the burning is often far below the actual temperature of ignition of these gases (firedamp 1,220°, carbonic oxide gas, 1,184°), on account of the absorption of the heat of ignition by the freely expanding gases.

900. Coal-Dust.—This discussion would not be complete without some special reference to the influence of this dangerous factor, present to a greater or less extent in many coal-mine explosions.

The presence of coal-dust suspended in the air of mine workings, and acted upon by a flame of sufficient volume and intensity, gives rise to *two* practical effects, viz.:

(*a*) Elongation and propagation of the flame.

(*b*) Widening of the explosive range of firedamp.

These effects have been described, (*a*), Art. **897** (*c*), and (*b*), Art. **860** (*b*).

The *facts* in regard to any kind of dust, and its influence upon the character of an explosion, are the following:

(*a*) The dust must be combustible, or it has, comparatively, no effect.

(*b*) The finer the dust and the more inflammable its nature, the quicker and fiercer will be its combustion.

(*c*) The free suspension of the dust in the air, and its complete combustion, are greatly assisted by a strong air current.

(*d*) The coal-dust (fine and larger particles) is heated to incandescence by the flame of the burning gases, distilling combustible gas, which adds to the flame, thereby transmitting the explosion through the airways.

(*e*) The incandescent carbon has the power to convert any carbonic acid gas (CO_2) with which it comes in contact into combustible carbonic oxide gas (CO).

The above facts are the results of practical experience, derived from actual observation of such occurrences, guided by an intelligent knowledge of the chemical possibilities, as demonstrated by experiment. The dust of anthracite coal is not susceptible of explosion under the prevailing conditions, being less friable and requiring a higher temperature to distil its gases.

901. Reducing Liability to Explosion.—The liability to accident by explosion can be reduced only by removing, as far as it is possible to do so, the causes and conditions which lead to such explosions. The incipient conditions of a mine explosion are, with rare exceptions, found in the following:

(*a*) A body of marsh-gas, collected in some cavity or recess of the roof or disused heading; or issuing suddenly from the working face, as a *feeder* or an *outburst*, and becoming transformed into a body of firedamp by its mixture with the air of the workings; and the presence and contact of the flame of a naked light, or a defective safety-lamp, or the projected flame of a blast, or, as sometimes occurs, the

flame of a safety-lamp blown through its gauze by the force of the current, or the force of a blast, to which it has been inadvertently exposed.

(*b*) The presence of a considerable quantity of fine coal, in the form of dust, in close proximity to a working face where blasting is performed; and the projection of the flame of a *blown-out shot* of such volume and intensity as to effect the raising of a cloud of the dust, and to convert the same into an incandescent volume generating combustible gas. This action has been proven, by the convincing results of experiments which leave no room to doubt, that the presence of marsh-gas, while it stimulates and strengthens the explosion, is by no means essential to it.

(*c*) The successive and quick firing of several shots, in a close working place, may precipitate an explosion, from the firing of considerable volumes of carbonic oxide gas, produced in the discharge of the first shots.

SAFETY-LAMPS.

DESCRIPTION OF LAMPS.

902. General Description.—A safety-lamp is a lamp of special construction. In appearance it very much resembles a small lantern, which it is. The flame is completely enclosed in wire gauze or in glass and wire-gauze casings, which prevent its contact with an outside body of gas. Its use serves two purposes; viz., *first*, protection in gaseous workings where an open light would cause serious results, by the ignition of the gas; and, *second*, to indicate to the miner the presence of gas.

903. Principle of the Safety-Lamp.—From our previous study of combustion (Art. **873**), we have learned that the temperature of the burning gases must not fall below the point of ignition of those gases, or the flame will be extinguished. Whenever this temperature is not reached at the initial points of reaction, there can be no ignition, and hence no flame.

In safety-lamps, the isolation of the flame is secured through the cooling effect of the wire gauze surrounding it. The gauze permits the passage of air or gas, but flame is extinguished when it comes in contact with the cool gauze. (See Art. **904.**)

904. Effect of Cooling.—Flame is the result of gases burning at a white heat. The temperature of ignition for each gas, in air, is a fixed point capable of calculation, and expresses the number of heat units evolved in the reaction, less the heat units absorbed and rendered latent by the products of the combustion. The proximity of any cooling surface whatever to a flame, has the effect of reducing the temperature of the reacting gases. The molecular vibrations of the cooler surface are so sluggish that the heat of the reaction in the flame is converted into *molecular work*, raising the temperature of the cool surface to a certain extent, but extinguishing the flame in its immediate proximity. This phenomenon may readily be observed by presenting a flame to a cool surface, when it will be seen that the flame does not touch the surface, but is separated by a thin layer of gas that does not burn, because it has been cooled below the point of ignition. This will continue as long as the surface remains cool. For the same reason, one may put a very cold hand, for a moment, into the flame of a fire without burning the hand or feeling the heat.

In the case of a flame impinging against a cool wire gauze, or other perforated surface, the conditions are very favorable for the cooling of the gases of the flame below the point of ignition, as they pass through the small openings. The gases are divided into minute streams or jets, by the meshes of the gauze, and cooled instantly, being thereby extinguished.

905. Temperature of Flame.—The cooling and extinguishing of a flame is greatly assisted by the air-currents pouring towards it, and diffusing among the gaseous molecules. This action of the air isolates, as it were, each burning hydrocarbon particle. Each separate particle is

thus surrounded by an envelope which renders more possible the cooling of the particle, because its temperature is somewhat below the temperature of ignition at the center. This has led to the rather indefinite and often misleading phrase, "*temperature of the flame.*" We can not rightly speak of the *temperature of flame* except in a general way, because it is not a definite quantity, but depends wholly upon conditions of which we have no gauge, and we find a different effective temperature in different parts of the same flame.

906. Requirements of a Good Lamp.—Safety-lamps, as previously stated, are used for two separate purposes, and this has given rise to two types of lamps, differing quite widely in their construction. They are:

(*a*) Lamps for general mining use.
(*b*) Lamps for testing for gas.

The requirements a good lamp must possess, for the purposes of general use in a mine, are the following:

(*a*) Safety in strong currents.
(*b*) Minimum liability to accident.
(*c*) Maximum illuminating power.
(*d*) Diffusion of light upwards.
(*e*) Simplicity of construction, and security of fastenings or lock.

907. Davy Lamp.—Fig. 115 shows a perspective view of this lamp. Fig. 116 is a sectional view of the same lamp. The Davy lamp admits air freely through the lower part of the gauze, as shown by the arrows at *a a*; while the products of combustion pass out through the upper portion of the gauze cylinder *b b* and the gauze plate *c* at the top of the lamp. This free passage of the gas-charged air in and out of the lamp ensures a good cap, and has made the Davy lamp a favorite with fire bosses, notwithstanding the danger that is always present in the unbonneted Davy lamp of the flame of the lamp being communicated to the outside gas, either through flaming in the lamp or from exposure

to a current. The lamp is not safe when exposed to a current of a greater velocity than 6 feet per second. When gas is present in a thin stratum at the roof, its presence will

FIG. 115. FIG. 116.

not be revealed by the Davy lamp. In the hands of a careful and experienced man, this lamp will detect the presence of gas in quantities as low as 3 per cent.

908. Stephenson Lamp.—This lamp consisted of a glass chimney surmounted by a perforated copper cap and surrounded by a perforated copper shield. The lamp gave a poor light, and was immediately supplanted by the Davy lamp with gauze covering.

909. Geordy Lamp.—This lamp, so called after George Stephenson, its inventor, was a combination of the glass chimney of the original Stephenson lamp and the gauze of the Davy lamp. It was regarded for a considerable time as a thoroughly reliable and safe lamp. It gave a better

light than the Davy lamp, and for a while came into quite extensive use. It was quite susceptible to gas, and made a good lamp for testing, because the gas-cap could be more easily distinguished through the glass than through the gauze, although the caps were not as high as in the Davy lamp. The supply, or feed, was more restricted, and entered the lamp below the flame and passed out through the gauze above the glass chimney.

910. Clanny Lamp.—This lamp was designed to secure greater protection for the flame, combined with a better light, than was provided in the Geordy lamp. A perspective view of the Clanny lamp is shown in Fig. 117, and a

FIG. 117. FIG. 118.

section of the same in Fig. 118. The air, instead of being admitted below the flame, as in the Geordy lamp, is admitted through the lower portion of the gauze cylinder, just above the glass, and descends, within the lamp, to the flame.

The lamp, while it may present some points of protection of the flame against strong currents, does not make a good lamp, either for testing or for general purposes of illumination. The glass is apt to become dimmed by the smoke of the flame, owing to the interference of the downward and upward currents above the flame. A considerable percentage of gas may be present in the air before its presence will be revealed by this lamp. The lamp loses its protective qualities whenever sufficient gas is present to produce flaming.

911. Evan Thomas Lamp.—This is an improvement upon the Clanny lamp, in two points; viz., the air drawn in at *a* (Fig. 119) is conducted downwards between the two

FIG. 119. FIG. 120.

glass chimneys with which the lamp is provided, and enters the lamp below the flame. The upper gauze of the lamp is provided with a sheet-iron bonnet, which is a great protection in case of flaming or inner explosion. The downward

current of cool air serves, also, to keep the glasses cool, and increases their power to transmit light; a heated glass always impairs the transmission of light.

Fig. 120 shows another type of this lamp, designed for the use of fire bosses. The inner glass cylinder is replaced by a cylinder of gauze; in other respects the principle of the lamps is the same. At the time of the invention of this lamp, the study and designing of lamps, with respect to securing greater protection, received renewed attention, and resulted in bringing forward various devices having this end in view.

912. Marsaut Lamp.—The principal feature of this lamp, a perspective view of which is shown in Fig. 121 and a section of the same in Fig. 122, is the multiple-gauze

FIG. 121. FIG. 122.

chimneys. The lamp shown in the figure (Fig. 122) has three of these gauze chimneys, one over the other, and an

outer bonnet of sheet iron. This lamp is adapted for use in strong currents. The air enters the lamp above the glass chimney; and much that has been said in reference to the Clanny lamp in this respect is applicable to the Marsaut.

913. Mueseler Lamp.—This lamp, of which Fig. 123 shows a perspective view and Fig. 124 a section, presents an

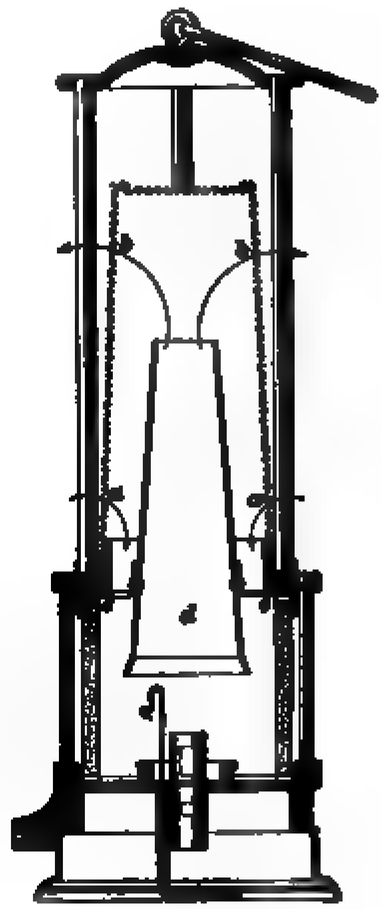


FIG. 123. FIG. 124.

important departure. It is the first lamp introducing a feature calculated to increase its draft, and thereby improve its illuminating power. In this lamp is provided a central tube or chimney *d* of sheet iron, conical in shape, and held in position by a horizontal, perforated diaphragm of sheet iron *e e*, at the junction of the gauze and glass cylinders. The air enters the lamp through the gauze at *a a*, and, passing through the perforations of the diaphragm, is drawn down under the expanded mouth of the central chimney and in close proximity to the flame. The draft of this chimney increases to a considerable degree the

illuminating power of the lamp, while the central tube adds very largely also to the security of the lamp against currents and inner explosions, the latter seldom being communicated outside of the lamp. It is not a lamp adapted to the detection of gas, but it has been known to withstand a current of 100 feet per second.

914. Howat's Deflector.—This consists of an annular ring A (Fig. 125), so arranged as to deflect the entering current of air downwards upon the flame. It has been fitted to the Marsaut lamp, with a marked improvement in the illuminating power of that lamp.

FIG. 125.

915. Ashworth-Hepplewhite-Gray Lamp.—Among the lamps of special design for testing for gas, that shown in Fig. 126 is perhaps the most convenient, and combines in one lamp many of the best features. The air, when the lamp is being used for testing, enters the tops of the four standards, as shown at *a a*, and, passing down the standards, enters the lamp below the flame, thereby producing the best conditions for yielding a good gas-cap. The glass chimney *c c* is made slightly conical, tapering towards the top; the same conical shape is, also, given to the gauze chimney *g*, above the glass. The gauze chimney is bonneted. The conical shape of the glass assists the

FIG. 126.

upward diffusion of the light, and makes the inspection of the roof easier; while the same shape in the gauze chimney renders the lamp more safe and secure against an inner explosion of gas being communicated outside of the lamp. The air being drawn into the lamp through the top of the standards, makes it possible with this lamp to detect a thin stratum of gas near the roof. When not in use for testing, the air may be admitted at the bottom of the standards, at *b b*, by moving a little shutter that closes them.

916. Pieler Lamp.—This lamp, shown in Fig. 127, is a gauze lamp, similar to a Davy lamp, but burns alcohol instead of oil, in order to render the observance of the gas-caps easier. In its safest form, the gauze is bonneted by a sheet-iron bonnet, the gas-caps being observed through a glass window. This window is very apt to become dimmed with smoke and moisture, and impair the observance of the caps. The lamp is provided with a shield *c*, surrounding the flame, and the latter is adjusted so that its tip does not extend above the top of the shield.

This lamp was designed by the inventor to yield a standard flame which would always present a certain height and volume, and yield flame-caps of a uniform height, for given percentages of gas. The following table was prepared by him to show the percentage of gas corresponding to different heights of flame-caps.

Fig. 127.

¼ per cent. of gas yields a cap 1.25 inches high.
½ per cent. of gas yields a cap 2.00 inches high.
1 per cent. of gas yields a cap 3.50 inches high.
1¼ per cent. of gas yields a cap 4.75 inches high.
1½ per cent. of gas { cap reaches the top of the lamp, and beyond this percentage of gas the lamp fills with flame.

The heights of these caps are measured from the top of the shield *c*. The lamp flames easily, in a mixture containing more than 1¾ per cent. of gas, and is, therefore, a source of danger, and requires great care and caution in its use. Any variation in the strength of the alcohol varies the height of the flame. The flame is, therefore, not strictly a standard flame.

917. The Illuminating Power of Safety-Lamps.—The amount of light given off by any safety-lamp is much less than that of the ordinary naked light used in mines. Table 22 gives the comparative illuminating power of some of the various lamps described. The light of a sperm candle is taken as 1, or unity.

TABLE 22.

Name of Lamp.	Illuminating Power of Lamp, with a Candle Taken as 1, or Unity.
Davy	0.16
Geordy	0.10
Clanny	0.20
Mueseler	0.35
Evan Thomas	0.45
Marsaut, 3 gauzes	0.45
Marsaut, 2 gauzes	0.55
Howat's Deflector	0.65
Ashworth-Hepplewhite-Gray	0.65 (about)

918. Flame-Caps or Gas-Caps.—By experiment, it has been determined that the presence of carbonic acid gas, even to the extent of 5 per cent., has no effect upon the flame-cap.

It has also been ascertained that the height of the flame-cap changes with the size and height of the flame itself, and also with the oil used to produce the flame.

919. The Oil Used in Safety-Lamps.—All the lamps described, with the exception of the improved Ashworth-Hepplewhite-Gray and the Pieler lamps, are constructed to burn either vegetable or seal oil. In the last two, light mineral oils are burned.

According to the English Mine Commission, the safest oils to use are vegetable oils, such as rape, made from rape-seed, and colza, made from cabbage-seed, and seal oil. None of these are explosive. Petroleum used alone is liable to explode, and should be avoided.

In point of brilliancy, the flame of a lamp burning seal oil is superior to one burning either rape or colza oil, and the wick is less liable to become charred.

By addition of one part of petroleum to two parts of rape or seed oil, the light is increased.

Many of the oils in common use in safety-lamps have a tendency to encrust the wick and thereby lower the flame. Sometimes petroleum or benzine has been added to the oil, which reduces this tendency, and yields a better flame for testing purposes. Alcohol yields a hotter and less luminous flame and a much higher cap. In some cases, a hydrogen flame has been used for testing purposes. The hydrogen is compressed into a small steel cylinder attached to the lamp, and is burned in the lamp at the mouth of a small tube. This apparatus gives a standard flame for testing, but it can not always be conveniently obtained.

920. Locks for Safety-Lamps.—All safety-lamps should be securely locked, and in such a manner as to preclude the possibility of being tampered with. Screw-pins are not an adequate protection.

The best lock, for security and cheapness, is the lead-plug lock, shown in Fig. 128.

On the right-hand side of the oil-vessel of the lamp a pin projects, with a hole in it. Around the bottom of the top part of the lamp there is a thin,

Fig. 128.

movable, metal ring *R*, and to this ring is fixed a hinged latch.

The ring is turned around, until the latch drops over the pin. A small plug of soft lead is put through the hole in the pin, to prevent the hinged latch from being lifted, and this lead plug is punched flat at both ends, to prevent it from being pulled out. The plugs are cast in a mold, at the colliery; and, as they are cut to pieces, in the lamp room, when the lamp is returned to be cleaned, they are collected and remelted, and the lead is used over and over again. To prevent tampering with the lead plug, it is punched up at both ends, with a punch containing a letter of the alphabet. These letters are interchangeable, and it is usual to use a new letter each day, so that the workmen can not counterfeit them.

Machines are used for locking these lamps, and other machines for cleaning them. Safety-lamps should be thoroughly cleansed at the close of every shift and put in readiness for another day.

TESTING FOR FIREDAMP IN MINES.

921. The Fire Boss.—The duties devolving upon a fire boss are of a very serious nature. In his hands is often placed the safety of every man in the mine. A simple oversight upon his part may result in the most appalling catastrophe.

The safety-lamp, at the present time, is the only practical means at the disposal of this man for the detection of firedamp. According to the good condition and sensitiveness of the lamp, and the experience of the man, his report of the condition of each working place and section of the mine under his charge is more or less accurate. That the fire boss should be a careful, painstaking, and conscientious man is readily seen upon a little reflection. Suppose, for example, a current of 50,000 cubic feet of air per minute is being furnished to a certain section of a gaseous mine. In this section, perhaps, the fire boss may detect a small percentage of gas in the current, say $\frac{1}{2}$ of one per cent. The

total flow of gas is then $\frac{1}{2}$ of $\frac{1}{100}$ of 50,000 = 250 cubic feet per minute. If a door is left open upon the airway, or a fall occurs so as to reduce the current, say, to 3,000 cubic feet of air per minute, this gas will render the reduced current explosive in a very short time, and only prompt and decisive action on the part of the fire boss will avert a catastrophe.

922. Testing by Lamp.—The use of the lamp, for the purpose of testing for gas, depends upon the observing of the height of a pale, bluish tip, or cap, to the flame of the lamp. If the lamp flame is too bright, a small gas-cap can not be seen, as the eye will be blinded by the light of the flame.

For this reason the non-luminous, alcohol or hydrogen flames are better adapted for observing the gas-cap. The body of the flame should be screened from the eye while taking an observation. This is sometimes effected by holding the hand between the flame and the eye, or by interposing a metallic screen, as in the Pieler lamp.

The flame of the lamp is usually lowered to a small size when testing, and it is always best to adopt a uniform size of flame, to ensure uniform results. No quick movement must be made. In case of flaming in the lamp, coolness and presence of mind are necessary to remove the lamp carefully from the gaseous body. A quick movement will precipitate an explosion by the forcing of the inner flame through the hot gauze.

A good lamp for testing purposes will have a free admission of air, preferably below the flame. The background of the flame, or the gauze through which the flame is observed, should present no reflecting surfaces, as any reflection interferes seriously with the sensitiveness of the observation.

The lamp is, thus far, the most practical means at our disposal for gas-testing in mines. The percentages of gas in the air, determined by its use, are necessarily only approximately accurate; but the determination is made at

once at the point where the gas has accumulated, and the value of this approximate knowledge can not be disputed.

923. Testing by Machines.—Undoubtedly the Shaw Gas-Testing Machine, a view of which is shown in Fig. 129, is the most accurate, simple, and complete mechanical device for this purpose known. Its use, however, is restricted largely by its lack of portability. On account of this, it can not replace the method of testing for gas, at the working faces, by means of the safety-lamp.

The machine consists of two cylinders, or pump-barrels, *A* and *B*, constructed of such relative size, and so connected

FIG. 129.

to a common lever *G*, as to pump relative quantities of gas and air into an ignition-chamber *Z*. One of the pump-cylinders *A* is stationary and pumps air, while the other cylinder *B* is so arranged as to be movable, and can be set to pump any proportionate amount of gas. Thus, it is easy to so arrange these two cylinders that a definite mixture of gas and air will be pumped into the ignition-chamber. The beam, or lever, is graduated to read the percentage of gas pumped.

The ignition-chamber *Z* is a cylinder having a loose piston. The mixture pumped into this chamber strikes first against

the piston, at the left-hand end of the cylinder, and, filling the cylinder, is expelled through an igniting nozzle at the opposite end. If the mixture is explosive, its ignition and explosion drives the piston forcibly against the gong *J*, at the end of the cylinder. The ignition is accomplished by a small gas-jet, or other flame burning at the discharge orifice of the chamber.

MINE VENTILATION.

(PART 1.)

INTRODUCTORY.

GRAVITATION.

924. As a knowledge of gravitation and the laws of falling bodies is necessary in the study of mine ventilation, these subjects will be briefly treated before the principles governing the flow of air are discussed.

925. All bodies in the universe exert a certain attractive force on every other body, which tends to draw the bodies together. This attractive force is called **gravitation.**

If a body is held in the hand, a downward pull is felt, and if let go of will fall to the ground. This pull is commonly called *weight*, but it really is the attraction between the earth and the body.

926. **Force of gravity** is a term used to denote the attraction between the earth and bodies upon or near its surface. It always acts in a straight line between the center of the body and the center of the earth. The force of gravity varies at points on the earth's surface.

It is slightly less on the top of a high mountain than at the level of the sea. For this reason the weight of a body also varies. But if the weight of a body at any place be divided by the force of gravity at that place, the result is called the *mass* of the body.

927. The **mass of a body** is the measure of the actual amount of matter that it contains, and is *always the same.*

If the mass of the body is represented by m, its weight

by W, and the force of gravity at the place where the body is weighed by g, we have

$$\text{mass} = \frac{\text{weight of body}}{\text{force of gravity}}, \text{ or } m = \frac{W}{g}. \qquad \textbf{(23.)}$$

928. Law of Gravitation:

The force of attraction by which one body tends to draw another body towards it is directly proportional to its mass, and inversely proportional to the square of the distance between their centers.

929. Laws of Weight:

Bodies weigh most at the surface of the earth. Below the surface, the weight decreases as the distance to the center decreases.

Above the surface, the weight decreases as the square of the distance increases.

ILLUSTRATION.—If the earth's radius is 4,000 miles, a body that weighs 100 pounds at the surface will weigh nothing at the center, since it is attracted in every direction with equal force. At 1,000 miles from the center it will weigh 25 pounds, since

$$4,000 : 1,000 = 100 : 25.$$

At 2,000 miles from the center it will weigh 50 pounds, since

$$4,000 : 2,000 = 100 : 50.$$

At 3,000 miles from the center it will weigh 75 pounds, and at the surface, or 4,000 miles from the center, it will weigh 100 pounds. If carried still higher, say 1,000 miles from the surface, or 5,000 miles from the center of the earth, it will weigh 64 pounds, since

$$5,000^2 : 4,000^2 = 100 : 64.$$

At 4,000 miles from the surface it will weigh 25 pounds, since

$$8,000^2 : 4,000^2 = 100 : 25.$$

930. Formulas for Gravity Problems:

Let W = weight of body at the surface;

w = weight of a body at a given distance above or below the surface;

d = distance between the center of the earth and the center of the body;

R = radius of the earth = 4,000 miles.

Formula for weight when the body is below the surface,

$$w R = d W. \qquad (24.)$$

Formula for weight when the body is above the surface,

$$w d^2 = W R^2. \qquad (25.)$$

EXAMPLE.—How far below the surface of the earth will a 25-pound ball weigh 9 pounds?

SOLUTION.—Use formula **24**, $w R = d W$.

Substituting the values of R, W, and w, we have

$$9 \times 4{,}000 = d \times 25;$$

or $$d = \frac{9 \times 4{,}000}{25} = 1{,}440 \text{ miles from the center. Ans.}$$

EXAMPLE.—If a body weighs 700 pounds at the surface of the earth, at what distance above the earth's surface will it weigh 112 pounds?

SOLUTION.—Use formula **25**, $w d^2 = W R^2$.

Substituting the values of R, W, and w, we have

$$112 \times d^2 = 700 \times 4{,}000^2;$$

or $$d = \sqrt{\frac{700 \times 4{,}000^2}{112}} = 10{,}000 \text{ miles.}$$

Therefore, 10,000 − 4,000 = 6,000 miles above the earth's surface. Ans.

EXAMPLE.—The top of Mt. Hercules was said to be 32,000 feet, say 6 miles, above the level of the sea. If a body weighs 1,000 pounds at sea-level, what would it weigh if carried to the top of the mountain?

SOLUTION.— $w d^2 = W R^2$; or, $w \times 4{,}006^2 = 1{,}000 \times 4{,}000^2$;

whence, $$w = \frac{4{,}000^2 \times 1{,}000}{4{,}006^2} = 997 \text{ pounds. Ans.}$$

EXAMPLES FOR PRACTICE.

1. How much would 1,000 tons of coal, weighed at the surface, weigh one mile below the surface? Ans. 1,999,500 lb.

2. How much would the coal in example 1 weigh one mile above the surface? Ans. 1,999,000 lb., nearly.

3 How far above the earth's surface would it be necessary to carry a body in order that it may weigh only half as much?

Ans. 1,656.854 miles, nearly.

4. A man weighs 160 pounds at the surface; how much will he weigh 50 miles below the surface? Ans. 158 lb.

5. If a body weighs 100 pounds 400 miles above the earth's surface, how much will it weigh at the surface? Ans. 121 lb.

NOTE.—Use 4,000 miles as the radius of the earth.

FALLING BODIES.

931. If a leaden ball and a piece of paper are dropped from the same height, the ball will strike the ground first. This is not because the leaden ball is the heavier, but because the resistance of the air has a greater retarding effect upon the paper than upon the ball. If we placed this same leaden ball and a piece of paper in a glass tube, Fig. 130, from which all of the air has been exhausted, it would be found that, when the tube was inverted, both would drop to the bottom in exactly the same time. This experiment proves that it was only the resistance of the air that caused the ball to reach the ground first in the former experiment. This resistance of the air may be nearly equalized by making the two bodies of the same shape and size. For example, if a wooden and an iron ball, having equal diameters, were dropped from the same height, they would strike the ground at almost exactly the same instant, although the iron ball might be ten times as heavy as the wooden ball.

FIG. 130.

Suppose there were several leaden balls, as shown in Fig. 131, at a; it is obvious that if they were dropped together, all would strike the ground at the same time. If the balls were melted together into one ball, as b, they would still fall together, and strike the ground in the same time as before.

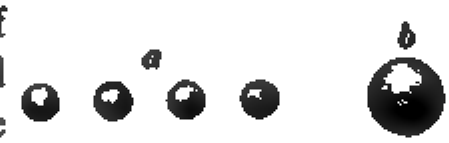


FIG. 131.

Since a number of horses can not run a mile in less time than a single horse, so 100 pounds can fall no farther in a given time than 1 pound can.

932. Acceleration is the rate of increase of velocity. If a force acts upon a body free to move, then, according to the first law of motion, it will move forever with the same velocity unless acted upon by another force.

Suppose that, at the end of one second, the same force were to act again, the velocity at the end of the second second would be twice as great as at the end of the first second. If the same force were to act again, the velocity at the end of the third second would be three times that at the end of the first second. So, if a constant force acts upon a body free to move, the velocity of the body at the end of any time will be the velocity at the end of one second, multiplied by the number of seconds.

This constant force is called a **constant accelerating force,** or **constant retarding force,** according as the velocity is constantly *increased* or *decreased.*

If a body is dropped from a high tower, the velocity with which it approaches the ground will be constantly increased or accelerated; for the attraction of the earth, or force of gravity, is constant and acts upon the body as a constant accelerating force. It has been found by careful experiments that this force of gravity, or constant accelerating force on a freely falling body, is equivalent to giving the body a velocity of 32.16 feet in one second; it is always denoted by g. As was mentioned before, g varies at different points on the earth, being 32.0902 at the equator and 32.2549 at the poles. Its value for this latitude (about 41° 25′ north) is very nearly 32.16, and this value should always be used in solving problems. It has also been found by experiment that a freely falling body starting from rest will have fallen 16.08 feet at the end of the first second; 64.32 feet at the end of the second second; 144.72 feet at the end of the third second; 257.28 feet at the end of the fourth second, etc., all of which are shown in the diagram, Fig. 132.

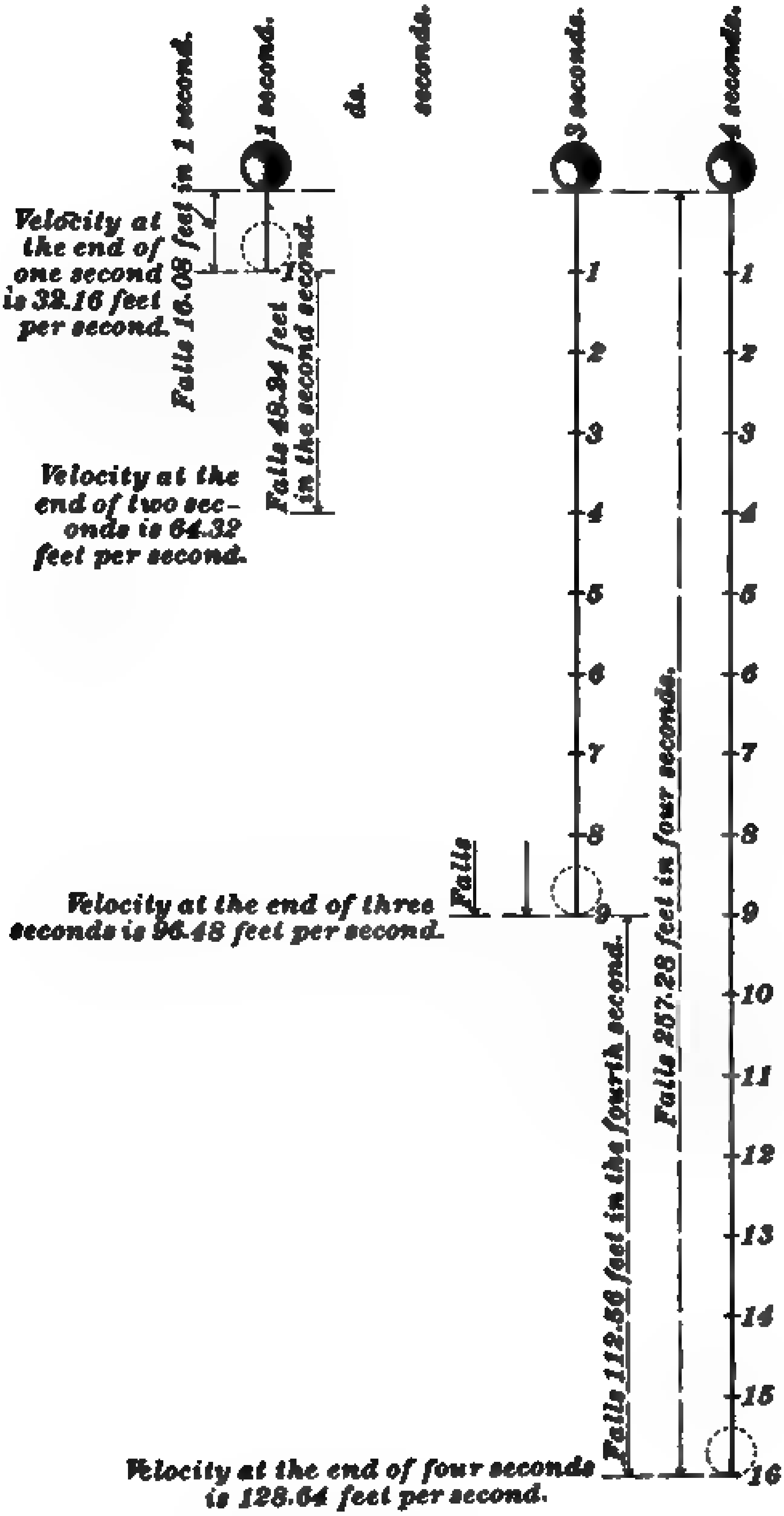

1 second.
2 seconds.
3 seconds.
4 seconds.
Velocity at the end of one second is 32.16 feet per second.
Falls 16.08 feet in 1 second.
Falls 48.24 feet in the second second.
Velocity at the end of two seconds is 64.32 feet per second.
Falls
Velocity at the end of three seconds is 96.48 feet per second.
Falls 112.56 feet in the fourth second.
Falls 257.28 feet in four seconds.
Velocity at the end of four seconds is 128.64 feet per second.

FIG. 138.

Since $\frac{64.32}{16.08} = 4 = 2^2$; $\frac{144.72}{16.08} = 9 = 3^2$; $\frac{257.28}{16.08} = 16 = 4^2$, and 2^2, 3^2, 4^2 are the squares of the number of seconds during which the body falls, it is easy to see that the space through which a body free to move will fall in a given time is equal to 16.08 multiplied by the square of the time in seconds.

Since $16.08 = \frac{32.16}{2} = \frac{1}{2} g$, the space $= \frac{1}{2} g \times$ square of time in seconds.

933. Formulas for Falling Bodies:

Let g = force of gravity = constant accelerating force due to the attraction of the earth;

t = number of seconds the body falls;

v = velocity at the end of the time t;

h = distance that a body falls during the time t.

$$v = g t. \qquad (26.)$$

That is, the velocity acquired by a freely falling body at the end of t seconds equals 32.16, multiplied by the time in seconds.

EXAMPLE.—What is the velocity of a body after it has fallen 4 seconds, assuming that the air offered no resistance ?

SOLUTION.—Using formula **26**,

$v = g t = 32.16 \times 4 = 128.64$ feet per second. Ans.

$$t = \frac{v}{g}. \qquad (27.)$$

That is, the number of seconds during which a body must have fallen to acquire a given velocity equals the given velocity in feet per second, divided by 32.16.

EXAMPLE.—A falling body has a velocity of 192.96 feet per second; how long had it been falling at that instant ?

SOLUTION.—Using formula **27**,

$t = \frac{v}{g} = \frac{192.96}{32.16} = 6$ seconds. Ans.

$$h = \frac{v^2}{2g}. \qquad (28.)$$

That is, the height from which a body must fall to acquire a given velocity equals the square of the given velocity, divided by 2 × 32.16.

EXAMPLE.—From what height must a stone be dropped to acquire a velocity of 24,000 feet per minute?

SOLUTION.— 24,000 ÷ 60 = 400 feet per second. Using formula **28**,

$$h = \frac{v^2}{2g} = \frac{400^2}{2 \times 32.16} = \frac{160,000}{64.32} = 2,487.56 \text{ feet. Ans.}$$

$$v = \sqrt{2gh}. \qquad \textbf{(29.)}$$

That is, the velocity that a body will acquire in falling through a given height equals the square root of the product of twice 32.16 and the given height.

EXAMPLE.—A body falls from a height of 400 feet; what will be its velocity at the end of its fall?

SOLUTION.—Using formula **29**,

$$v = \sqrt{2gh} = \sqrt{2 \times 32.16 \times 400} = 160.4 \text{ feet per second. Ans.}$$

$$h = \tfrac{1}{2}gt^2. \qquad \textbf{(30.)}$$

That is, the distance a body will fall in a given time equals 32.16 ÷ 2, multiplied by the square of the number of seconds.

EXAMPLE.—How far will a body fall in 10 seconds?

SOLUTION.—Using formula **30**,

$$h = \tfrac{1}{2}gt^2 = \tfrac{1}{2} \times 32.16 \times 10^2 = 1,608 \text{ feet. Ans.}$$

$$t = \sqrt{\frac{2h}{g}}. \qquad \textbf{(31.)}$$

That is, the time it will take a body to fall through a given height equals the square root of twice the height, divided by 32.16.

EXAMPLE.—How long will it take a body to fall 4,116.48 feet?

SOLUTION.—Using formula **31**,

$$t = \sqrt{\frac{2 \times 4,116.48}{32.16}} = 16 \text{ seconds. Ans.}$$

A body thrown vertically upwards starts with a certain velocity called the **initial velocity.** In this case gravity acts as a constant retarding force. The formulas given above will also apply in this case.

Example.—If a cannon-ball is shot vertically upwards with an initial velocity of 2,000 feet per second, (*a*) how high will it go? (*b*) How long a time must elapse before it reaches the earth again?

Solution.—(*a*) Using formula **28**,

$$h = \frac{v^2}{2g} = \frac{2{,}000^2}{2 \times 32.16} = 62{,}189 \text{ feet, nearly,} = 11.778 \text{ miles. Ans.}$$

To find the time it takes to reach a height of 62,189 feet, use formula **27**.

$$t = \frac{v}{g} = \frac{2{,}000}{32.16} = 62.19 \text{ seconds.}$$

Since it will take the same length of time to fall to the ground, the total time will be $62.19 \times 2 = 124.38$ seconds $= 2$ minutes 4.38 seconds. Ans.

THE NECESSITY OF VENTILATION.

934. Ventilation is the replacing of the foul air contained in an enclosed space by fresh air from the atmosphere.

To a person accustomed to working out of doors the necessity of ventilation is not apparent. He breathes, and the foul gas exhaled from his lungs dissipates into the ocean of atmosphere about him, leaving no trace behind, so rapidly is it diluted by the ever-moving air around him. When he descends into a mine, the case is widely different. Here, unless assisted by artificial means, the air-currents move very slowly or not at all. Poisonous gases from the workings must be diluted by fresh air; the men require a certain amount of fresh air to sustain life; the lamps require a certain amount in order that they may burn and give forth light; the horses or mules require still more; air or an air-current is required for other purposes. The result of all this is that unless a constant supply of fresh air is being circulated through the mine, it very soon becomes impossible for men or animals to live in it—much less work there.

The science of mine ventilation may be comprised under three general headings:

1. The quantity of air required.
2. The laws governing the flow of air through mines.
3. The means for inducing the flow of air through mines.

THE QUANTITY OF AIR REQUIRED.

935. The question as to what amount of air is necessary in mines does not admit of an exact answer. No two mines present the same conditions, and what is an ample provision of air in one mine is inadequate in another.

As each man requires a certain amount of pure air at every breath, it has been the rule in the past to select one man as the unit of calculation, and to allow so many cubic feet for every man employed underground. Some writers have made additional allowances for the mules and lamps.

Any estimate based on these lines is mere guesswork. The amount of air necessary for the support of life and the combustion of lights is insignificant in comparison with the other requirements.

A man requires a quantity of air which varies according to the exertion he is making, and this quantity, for a miner, may be estimated at 28 cubic feet per hour, or half a cubic foot per minute. A lamp consumes about the same quantity, and a mule about six times as much as a man.

A considerable quantity of air is required to render harmless the gas transpiring from the coal. If this gas were given off regularly, a correct estimate of the quantity of air required to dilute and render it harmless could be arrived at; but, owing to sudden outbursts, this can not be done.

A shallow mine is more likely to have had the gas drained off by the nearness of the seam to the surface, and is, therefore, not likely to require so much air for the removal of gas, in working, as a deep mine.

A change in the barometer has a decided influence upon the ventilation. A low barometer indicates a lighter weight of the air, and this, by reducing the pressure, assists in the freer admission of standing gas from the goaves and disused workings, and makes necessary an increased quantity of air to remove this gas. Heated air requires more fresh air to reduce the temperature and make the atmosphere of the mine healthy and comfortable for the workmen.

At the same time, it is necessary to remember that,

although the current should be sufficiently strong to enable it to be felt if the face is turned towards it, it must still not be so strong as to chill those who enter it while sweating.

936. The laws relating to the ventilation of coal-mines in the different States of the Union require, with two exceptions, a minimum of 100 cubic feet of air per man per minute. The Anthracite Mine Law of Pennsylvania fixes 200 cubic feet per man per minute as the minimum. The law of the State of Maryland fixes no minimum, but requires that the mine "shall be in a healthful condition for the men working therein." The English Mines Regulation Act of 1887 requires "sufficient to dilute and render harmless all noxious gases."

937. Instead of attempting to fix the quantity required at so much per man, it would be better to class the mines in each district into groups, having reference to the number of men employed, the area of the workings, the output, the nature of the coal, the depth of the workings from the surface and the general conditions regarding the amount of gas evolved, etc., and to make an average estimate of the volume required for the mines of each group.

In such a classification, the increase of the ventilation would be in accordance with the importance of the different requirements. These requirements may be summarized as follows:

The total quantity of air required should increase—

1. With the maximum number of men employed;
2. With the maximum number of mules in use;
3. With the maximum quantity of explosives used;
4. With the maximum daily output;
5. With the depth of the seam from the surface;
6. With the thickness of the seam;
7. With the extent of the live workings;
8. With the extent of the gob.

The volume of air to be allowed for these causes can be determined only after careful and exhaustive research, but,

if determined, it would ensure safety much more certainly than the minimum system at present in vogue.

In the largest and most gaseous mine in the anthracite region of Pennsylvania, the average quantity provided per man per minute ranges from 200 to 700 cubic feet.

THE LAWS GOVERNING THE FLOW OF AIR.

THE THEORETICAL VELOCITY OF AIR.

938. The **theoretical velocity** of air is the velocity at which the air enters the downcast shaft, and before it is subject to the resistance of friction due to the sides of the mine passages. It is a purely theoretical quantity and of little practical use. To produce a flow of air between the upcast and downcast shafts, the pressure, or weight, of the column of air in the downcast must be greater than the pressure, or weight, of the column of air in the upcast.

939. Suppose that Fig. 133 represents a section of a mine in which the downcast shaft *A B* and the upcast shaft *D C* have the same height. The air can be caused to flow from *A* to *D* by creating a difference of pressure or of weight in the columns of air in the two shafts, that in the shaft *D C* being less than that in the shaft *A B*.

FIG. 133.

This difference of pressure, or weight, of the air columns can be created in two ways: (1) By increasing the density, or pressure, of the air in the shaft *A B*. (2) By expanding the air, or decreasing the pressure, in the shaft *D C*. Each of these methods results in destroying the equilibrium, or equality of pressure, or weight, in the shafts.

Without entering here into a description of the methods for producing the difference of pressure, it may be stated that the ventilation is accomplished, according to the first of the above methods, by the use of a blowing-fan or by a waterfall, and, according to the second method, by means of a furnace, exhaust-fan, or steam-jet.

940. To find the theoretical velocity of air in a mine, due to the difference in the pressures in the upcast and downcast shafts, we have the following formula, in which

v = velocity of the air in feet per second;
F = the constant force represented by difference of pressure in pounds per square foot;
w = weight of a cubic foot of air;
g = acceleration due to gravity = 32.16 ft.

$$v = \sqrt{\frac{2gF}{w}}. \qquad (32.)$$

941. The Motive Column.—That portion of the downcast column of air which represents the difference between the weights of the air columns in the downcast and the upcast shafts is called the **motive column.** The excess of weight in the air in the downcast over that in the upcast is what causes the flow of air up the upcast. Hence, if we subtract the pressure per square foot at the bottom of the upcast from the pressure per square foot at the bottom of the downcast, and divide the difference by the weight of a cubic foot of air in the downcast, we have the length of the motive column, or the column whose weight overcomes the balance and causes the current to move up the upcast. For example, in

FIG. 134.

Fig. 134, the long hot column ed is equal in weight to the short cold column bc, and they balance each other, but the column ab of cold air is resting on bc and destroys the balance, and causes a current to flow; hence, ab is the motive column.

942. The length of the motive column may be found by means of one of the following formulas, in which

W = the weight of a cubic foot of air in the downcast shaft;
p = the pressure of the downcast shaft;
p_1 = the pressure in the upcast shaft;
t_1 = the average temperature of the air in the downcast shaft;
t = the average temperature of the air in the upcast shaft;
D = the depth of the upcast shaft in feet;
M = the length of the motive column in feet;
G = the water-gauge in inches. (See Art. **1058.**)

Then,
$$M = \frac{p - p_1}{W}. \qquad \textbf{(33.)}$$

$$M = \frac{5.2\,G}{W}. \qquad \textbf{(34.)}$$

$$M = \frac{D(t - t_1)}{459 + t}. \qquad \textbf{(35.)}$$

EXAMPLE.—If the temperature of the air in the downcast shaft is 40° F., and in the upcast shaft 120° F., what is the height of the motive column, the depth of the upcast shaft being 200 feet?

SOLUTION.—Applying formula **35**,

$$M = \frac{D(t - t_1)}{(459 + t)} = \frac{200(120 - 40)}{(459 + 120)} = 27.63 \text{ feet.}$$

PROOF.—The proof of the accuracy of this conclusion may be found as follows: The weight of a cubic foot of air in the downcast shaft is .07968, and the weight of a cubic foot of air in the upcast shaft is .06867; then the entire weight of the upcast shaft column is $.06867 \times 200 = 13.73400$ pounds, and the weight of the portion of the downcast column that balances the weight of the upcast column is $200 - 27.63 = 172.37$, and $172.37 \times .07968 = 13.734$ pounds.

In formula **35** it is assumed that the temperatures of the downcast and outer air are the same. There is no material error involved in this assumption, and the height of the motive column so obtained is practically correct, since any increase in temperature due to the depth is partly, if not wholly, neutralized by the moisture in the shaft absorbing heat from the air.

PRESSURE AND RESISTANCES.

943. When the word "pressure" is used in mine ventilation, it means the force that produces a movement of the air through the workings, and is called the **ventilating pressure.** The velocities of air-currents depend upon differences in pressures, the greater the difference the greater the velocity of the current. It should, therefore, be remembered that it is not the gross pressure at the beginning of an air-current that produces its velocity, but rather the difference between the gross pressures at both ends, which is the *ventilating pressure.* A difference of pressure of one pound per square foot will produce a current of wind in the open air having a velocity of about 19 miles per hour.

944. The ventilating pressure may be expressed in pounds (in which case it is called the **total pressure**), in pounds per square foot, in inches of water-gauge, or in feet of motive column. Unless otherwise stated, it will be expressed in pounds per square foot. Should it be necessary to express it in inches of water-gauge, it may be easily converted into pounds per square foot by multiplying the number of inches of water-gauge by 5.2.

In order to avoid the long term "ventilating pressure,' and also to make the language conform to other books pertaining to the subject of mine ventilation, the word *pressure* only will be used, except when it is thought best to use the full term.

The resistances met with in mines may be divided into three classes: First, the resistance due to friction; second, the resistance due to changing the direction of the current,

i. e., bends; third, the resistance due to contracting or enlarging the airway.

The most important of these resistances is that due to friction, and is the first that will be considered.

THE THREE LAWS OF FRICTION.

945. As the result of many experiments, the truth of the three following laws, called the **three laws of friction,** has been firmly established.

946. First Law.—*When the velocity remains the same, the total pressure required to overcome friction varies directly as the extent of the rubbing surface.*

947. By **rubbing surface** is meant the entire area touched by the air in passing through the airway. The cross-section of the airway may be a square, a rectangle, a trapezoid, or a circle. If the cross-section is a square, the perimeter equals the length of one of the sides of the cross-section multiplied by 4; if a rectangle or trapezoid, the perimeter equals the sum of all the sides, and if a circle, the perimeter equals the diameter multiplied by 3.1416. Having found the perimeter, the rubbing surface may be found by multiplying the perimeter by the length of the airway.

948. The first law states that if the rubbing surface be increased, the pressure must be increased in the same proportion in order to pass the air with the same velocity. In other words, if the rubbing surface be increased $1\frac{1}{2}$, 2, 3, 4, etc., times, the pressure must also be increased $1\frac{1}{2}$, 2, 3, 4, etc., times in order to pass the same quantity of air.

In applying this law, it does not matter whether the pressure per square foot or the total pressure is considered, if in the first case the sectional area remains the same.

EXAMPLE.—Suppose that a certain airway passes 10,000 cubic feet of air per minute; what must be the increase in pressure in order to pass the same amount through an airway whose cross-section has the same area, but whose rubbing surface is 1.6 times as great?

SOLUTION.—Since the rubbing surface is increased 1.6 times, while the other factors (velocity, quantity, sectional area, etc.) remain the same, it follows that, according to the first law of friction, the pressure must also be increased 1.6 times. Ans.

949. The form of the cross-section of the airway exerts a considerable influence on the amount of rubbing surface, as the following examples will show:

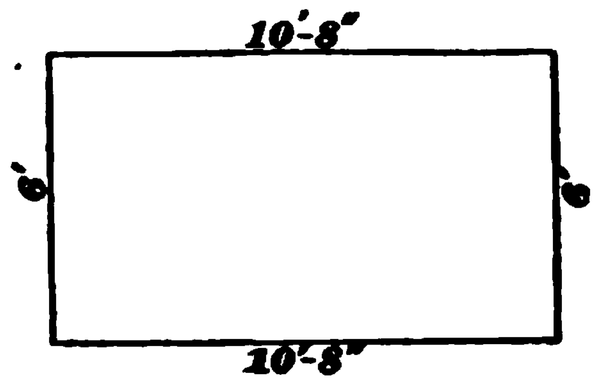


FIG. 135.

EXAMPLE 1.—Find (*a*) the rubbing surface and (*b*) the area of the cross-section of an airway 1,000 feet long having a rectangular cross-section, whose sides are 10 feet 8 inches long by 6 feet high. (See Fig. 135.)

SOLUTION.—(*a*) The rubbing surface equals the perimeter multiplied by the length; or, since 10 ft. 8 in. $= 10\frac{2}{3}$ ft., $(10\frac{2}{3} + 6 + 10\frac{2}{3} + 6) \times 1{,}000 = 33\frac{1}{3} \times 1{,}000 = 33{,}333\frac{1}{3}$ sq. ft. Ans.

(*b*) Area $= 10\frac{2}{3} \times 6 = 64$ sq. ft. Ans.

EXAMPLE 2.—Suppose that in the preceding example the rectangular section had been 16 feet wide and 4 feet high, what would have been the rubbing surface and area?

SOLUTION.—The rubbing surface $= (16 + 4 + 16 + 4) \times 1{,}000 = 40 \times 1{,}000 = 40{,}000$ sq. ft., and the area $= 16 \times 4 = 64$ sq. ft. Ans.

In this example the cross-sectional area is the same as in example 1, while the rubbing surface is $\frac{1}{5}$ greater. Had the sides been 32 feet and 2 feet, the sectional area would have been 64 square feet, as above, but the rubbing surface would have been $(32 + 2 + 32 + 2) \times 1{,}000 = 68{,}000$ square feet, or 2.07 times as much as in example 1. Hence, to pass the same quantity of air, the pressure would require to be increased 1.07 times.

950. It is easy to see that the more oblong the rectangle is the more rubbing surface there is for the same sectional area, and it is evident that the perimeter of a square section is less than that of a rectangular section having the same area. Thus, the perimeter of the square, Fig. 136, is but 32 feet, while that of the rectangle in Fig. 135 is $33\frac{1}{3}$ feet, and that of the rectangle in example 2 is 40 feet. If

Fig. 136 represents a section of an airway 1,000 feet long,

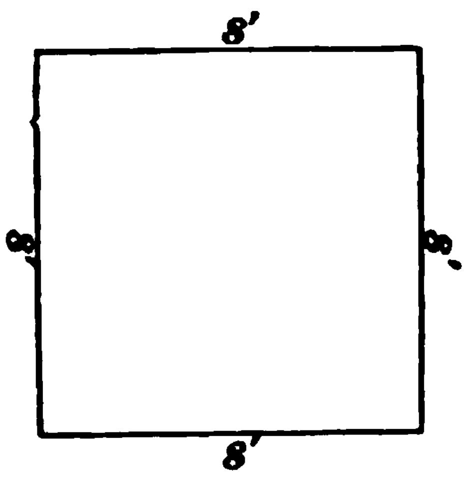


FIG. 136.

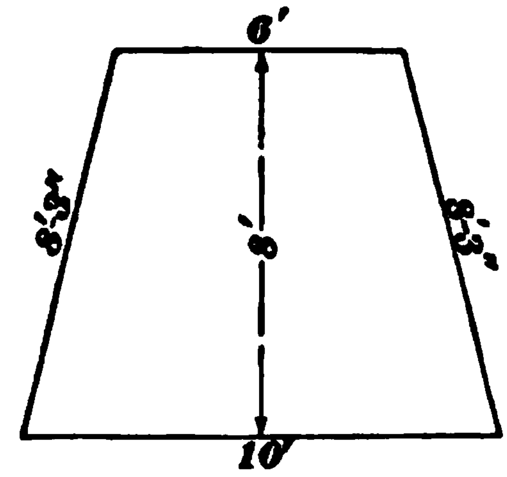


FIG. 137.

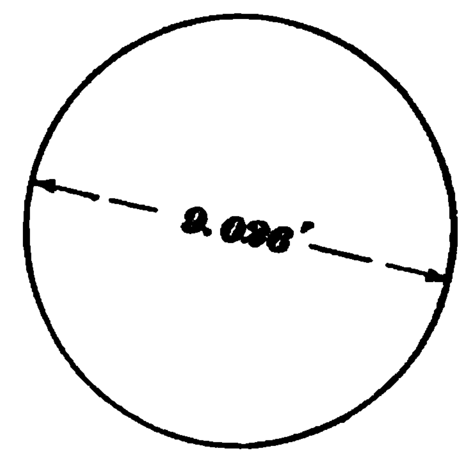


FIG. 138.

the rubbing surface is $32 \times 1{,}000 = 32{,}000$ square feet.

EXAMPLE 3.—Suppose an airway to have a trapezoidal cross-section like that shown in Fig. 137, and to be 1,000 feet long; what is the rubbing surface and sectional area?

SOLUTION.—The rubbing surface (since 8 ft. 3 in. $= 8\frac{1}{4}$ ft.) equals $(10 + 8\frac{1}{4} + 6 + 8\frac{1}{4}) \times 1{,}000 = 32{,}500$ sq. ft., and sectional area $= \frac{6 + 10}{2} \times 8 = 64$ sq. ft. Ans.

EXAMPLE 4.—A circular airway is 9.026 feet in diameter and 1,000 feet long; what is its rubbing surface and sectional area?

SOLUTION.—The rubbing surface equals 3.1416 times the diameter multiplied by the length $= 3.1416 \times 9.026 \times 1{,}000 = 28.36 \times 1{,}000 = 28{,}360$ sq. ft., and the sectional area $= 9.026^2 \times .7854 = 64$ sq. ft.

It will be noticed that in all of the above examples the area of the cross-section is 64 square feet, while the rubbing surface varies from 28,360 to 68,000 square feet.

The results obtained above are all combined in the following table, and show at a glance the effect produced by varying the forms of the cross-section, the sectional area and the length of the airway being the same in all.

951. Table 23 shows that for a given sectional area the circular airway has the least rubbing surface, and that the square airway comes next, while, with rectangular airways, that which most nearly approaches the square form has the least rubbing surface. Hence, for economy in ventilation, circular airways are best; but, since they are seldom practicable, owing to other considerations, square airways should be used whenever it is possible to do so. If rectan-

gular or trapezoidal airways are absolutely necessary, they should, in so far as it is practicable, approach the square form.

TABLE 23.

Form of Section.	Dimensions of Section.	Length in Feet.	Perimeter in Feet.	Rubbing Surface in Square Feet.	Sectional Area in Square Feet.
Circular	9.026′ diam.	1,000	28.36	28,360	64
Square	8′ × 8′	1,000	32	32,000	64
Trapezoidal	(10′ and 6′) × 8½′	1,000	32½	32,500	64
Rectangular	10′ 8″ × 6′	1,000	33⅓	33,333	64
Rectangular	16′ × 4′	1,000	40	40,000	64
Rectangular	32′ × 2′	1,000	68	68,000	64

It is evident that increasing the length of the airway will also increase the rubbing surface. Hence, if two airways have the same perimeter and area, but different lengths, and the air is transmitted with the same velocity in each, the pressures will be in direct proportion to the lengths.

EXAMPLE.—Two airways are each 6 feet high and 9 feet wide; consequently their areas of section and perimeters are equal. The length of one of these airways is 1,800 feet, and the pressure indicated by the water-gauge is 1.72 inches; the length of the other airway is 2,700 feet. If the velocity is the same in both these airways, what should be the height of the water-gauge for the airway 2,700 feet in length?

SOLUTION.—Since the areas, perimeters, and velocities are the same, the heights of the water-gauges will be directly as the lengths of the airways, or $1.72 : x :: 1{,}800 : 2{,}700$; whence, $x = 2.58$ in. Ans.

952. Second Law.—*When the velocities and rubbing surfaces remain the same, the pressures required to force air through the passages of a mine increase and decrease inversely as the sectional areas of the passages increase or decrease.*

953. The second law states that if the velocity remains the same and the rubbing surfaces are equal, the pressure per square foot will increase as the sectional area decreases;

or, the pressure per square foot will decrease as the sectional area increases; that is, if the sectional area be reduced to $\frac{1}{2}$, $\frac{1}{4}$, $\frac{1}{6}$, $\frac{1}{10}$, etc., of the original sectional area, the pressure per square foot must be increased, 2, 4, 6, 10, etc., times, respectively, to pass the air with the same velocity, the rubbing surface being the same in both cases. Or, if the sectional area be increased 2, 4, 6, 10, etc., times the original sectional area, the pressure per square foot may be, respectively, reduced to $\frac{1}{2}$, $\frac{1}{4}$, $\frac{1}{6}$, $\frac{1}{10}$, etc., of the original pressure per square foot required to pass the air with the same velocity, the rubbing surface being the same in both cases.

EXAMPLE.—Suppose that the pressure per square foot required to pass air at a given velocity is .02 inch per square foot in a square airway 8 feet high and 8 feet wide. What pressure per square foot will be required to pass air at the same velocity through a circular airway whose perimeter is the same as that of the square one, namely, 32 feet, and whose length is the same as that of the square one?

SOLUTION.—According to the second law of mine friction, when the velocities are the same, the pressures vary inversely as the sectional areas of the airways. If the perimeter be divided by 3.1416, the quotient will be the diameter of the section of the circular airway; and the square of this diameter multiplied by .7854 is the area of the cross-section of the circular airway in square feet; hence,

$$\text{area} = \left(\frac{32}{3.1416}\right)^2 \times .7854 = 81.49 \text{ square feet.}$$

Then the pressure required is found by the proportion $81.49 : 64 :: .02 : x$; or, $x = .0157$ inch of water-gauge. Ans.

954. The second law only applies to *pressure per square foot*, for the total pressure remains the same, as it should; since, when the rubbing surface and velocity remain the same, the total resistance (= total pressure) must also remain the same, no matter what the sectional area may be. Thus, in the above example, the total pressure for the square airway is $64 \times .02 = 1.28$ lb., and for the circular airway, $81.49 \times .0157 = 1.28$ lb.

EXAMPLE.—(*a*) An $8' \times 10'$ rectangular airway is 5,000 feet long; what must be the length of a similar airway, $6' \times 8'$, having the same rubbing surface? (*b*) If a pressure of .5 pound per square foot is required to pass the air through the $8' \times 10'$ airway with a certain velocity, what pressure per square foot is required to pass the air through the $6' \times 8'$ airway with the same velocity?

SOLUTION.—(*a*) The rubbing surface of the $8' \times 10'$ airway is $(8 + 10 + 8 + 10) \times 5,000 = 180,000$ sq. ft. The perimeter of the $6' \times 8'$ airway is $6 + 8 + 6 + 8 = 28$ ft. Consequently, the length of the $6' \times 8'$ airway is $180,000 \div 28 = 6,428\frac{4}{7}$ ft. Ans.

(*b*) Since, according to the second law of friction, the pressures per square foot vary inversely as the sectional areas, when the rubbing surfaces and velocities remain the same, and the sectional areas are $8 \times 10 = 80$ sq. ft., and $6 \times 8 = 48$ sq. ft., $80 : 48 :: x : .5$; or, $x = .8\frac{1}{3}$ lb. per square foot. Ans.

Here again the total pressures are the same, since $80 \times .5 = 40$ lb., and $48 \times .8\frac{1}{3} = 40$ lb.

955. Third Law.—*The pressure required to overcome friction in an airway varies as the squares of the velocities when the rubbing surface and the areas of section are the same; and the pressures required to overcome friction vary as the squares of the velocities multiplied by the rubbing surfaces per square foot of section in all airways.*

956. If the velocity be increased $1\frac{1}{2}$, 2, 3, 5, etc., times, the rubbing surface remaining the same, the pressure must be increased $(1\frac{1}{2})^2$, 2^2, 3^2, 5^2, etc., or $2\frac{1}{4}$, 4, 9, 25, etc., times, respectively; and if the velocity be reduced $1\frac{1}{2}$, 2, 3, 5, etc., times, the rubbing surface remaining the same, the pressure must be reduced $(1\frac{1}{2})^2$, 2^2, 3^2, 5^2, etc., or $2\frac{1}{4}$, 4, 9, 25, etc., times, respectively.

If the sectional area and rubbing surface both remain the same, the pressure per square foot will also vary directly as the square of the velocity.

EXAMPLE.—Suppose that in the last example the velocity was 400 feet per minute, and that it was desired to increase it to 450 feet per minute, what would be the total pressure required?

SOLUTION.—Since the pressures vary directly as the squares of the velocities, $400^2 : 450^2 :: 40 : x$; or, $x = 50\frac{5}{8}$ lb. Ans.

EXAMPLE.—In the above example, what would be the pressure per square foot, were the velocity increased from 400 to 450 feet per minute in the $6' \times 8'$ airway?

SOLUTION.—The pressure per square foot was found to be $.8\frac{1}{3}$ pound; hence, according to the third law, since the sectional area and rubbing surface remain the same, $400^2 : 450^2 :: .8\frac{1}{3} : x$; or $x = 1.055$ lb. per square foot. Ans.

THE COEFFICIENT OF FRICTION.

957. By means of the three laws of friction, and by the aid of other laws which can be deduced from them, and which will be given later, it is possible, when all of the data for one airway and a part of the data for another airway are known, to calculate the remaining data for the second airway; or, if all the data for an airway are known, to calculate the effect produced by varying the pressure, velocity, etc. But in order to calculate the pressure required to force the air (overcome the resistances) through a given airway, to calculate the pressure required to pass a certain quantity per minute through a given airway, and to calculate the horsepower, etc., it is necessary to know the *coefficient of friction.*

958. *The* ***coefficient of friction*** *is that amount of the total ventilating pressure which is required to overcome the resistance offered by one square foot of rubbing surface when the velocity is 1 foot per minute.*

959. For example, this may be further explained by stating that the coefficient of friction is equivalent to the pressure required to overcome the friction in an airway one-quarter of a foot long, 1 foot square in section, and through which the air is passing with a velocity of 1 foot per minute.

Since the total pressure may be expressed in pounds, or as so many feet of motive column, having a cross-section equal to the sectional area of the airway, the coefficient of friction may also be expressed as a fraction of a pound or a factor of the motive column in feet. In the various works treating on mine ventilation, the coefficient is usually expressed in pounds, and will be so expressed throughout this discussion.

The coefficient of friction then becomes a unit which, multiplied by the rubbing surface in square feet (according to the first law), and again multiplied by the square of the velocity in feet per minute (according to the third law), will give the total ventilating pressure in pounds.

960. The coefficient of friction varies somewhat for different mines, according to the degree of smoothness of the rubbing surface, and probably to a slight extent on account of the character of the material forming the sides of the airway. Different experimenters have obtained values which show considerable variation in their results; but the value most commonly used is that determined by J. J. Atkinson, and is the one which will be used in this discussion. This unit is known to be too high, but since every change in direction, owing to bends, and every reduction or enlargement of the passageway, etc., entails extra losses which are very difficult to calculate, it will be more convenient to use Atkinson's coefficient and disregard the extra losses. By so doing, the entire air-course is treated as if it were a straight airway, and the calculations are greatly simplified. The value of Atkinson's coefficient of friction is .0000000217 pound. In other words, the pressure required to overcome the resistance offered by 1 square foot of rubbing surface when the velocity is 1 foot per minute, is that part of a pound represented by 217 divided by 1 followed by 10 ciphers, or .0000000217, expressed decimally.

961. Example.—(*a*) What is the total pressure required to overcome the frictional resistances of a $6' \times 8'$ airway, 12,750 feet long, if the velocity is 480 feet per minute? (*b*) What is the pressure per square foot? (*c*) What should the water-gauge read?

Solution.—(*a*) According to the foregoing statements, the total pressure is equal to the continued product of the coefficient of friction, the rubbing surface, and the square of the velocity; hence, since the rubbing surface $= 28 \times 12{,}750 = 357{,}000$ sq. ft., total pressure $= .0000000217 \times 357{,}000 \times 480^2 = 1{,}784.89$ lb. Ans.

(*b*) The pressure per square foot equals the total pressure divided by the sectional area $= \frac{1{,}784.89}{8 \times 6} = 37.18$ lb. per square foot. Ans.

(*c*) Since 1 inch of water-gauge represents a pressure of 5.2 pounds per square foot, 37.18 pounds represent $\frac{37.18}{5.2} = 7.15$ in. Ans.

962. To express the foregoing by means of formulas, let

$P =$ total ventilating pressure in pounds;

$p =$ ventilating pressure in pounds per square foot;

a = sectional area of airway in square feet;
k = coefficient of friction = .0000000217;
s = total rubbing surface in square feet;
v = velocity of air in airway in feet per minute;
o = perimeter of airway in feet;
l = length of airway in feet;
W = water-gauge in inches.

Throughout this subject the letters as printed above will always represent the same quantities.

$$P = p\,a. \qquad \textbf{(36.)}$$

That is, the total pressure equals the pressure per square foot multiplied by the sectional area of the airway.

EXAMPLE.—If the sectional area of the airway is 56 square feet, and the pressure per square foot is 8.46 pounds, what is the total pressure?

SOLUTION.—Applying formula **36**,

$$P = p\,a = 8.46 \times 56 = 473.76 \text{ lb. Ans.}$$

$$P = k\,s\,v^2. \qquad \textbf{(37.)}$$

That is, the total pressure equals the continuea product of the coefficient of friction, the rubbing surface, and the square of the velocity.

EXAMPLE.—An airway 6′ × 6′ and 5,000 feet long passes air with a velocity of 340 feet per minute; what is the total ventilating pressure?

SOLUTION.—Applying formula **37**, $s = 6 \times 4 \times 5{,}000 = 120{,}000$ sq. ft., and $v = 340$. Hence, $P = k\,s\,v^2 = .0000000217 \times 120{,}000 \times 340^2 =$ 301 lb., nearly. Ans.

$$p = \frac{k\,s\,v^2}{a}. \qquad \textbf{(38.)}$$

That is, the pressure per square foot equals the continued product of the coefficient of friction, the rubbing surface, and the square of the velocity, divided by the sectional area of the airway.

EXAMPLE.—What is (*a*) the pressure per square foot in the last example? (*b*) the water-gauge?

SOLUTION.—(*a*) Substituting in formula **38**, $a = 6 \times 6 = 36$ sq. ft., and

$$p = \frac{.0000000217 \times 120{,}000 \times 340^2}{36} = 8.36 \text{ lb. Ans.}$$

(*b*) Since $p = 5.2\,W$, $W = \frac{p}{5.2} = \frac{8.36}{5.2} = 1.61$ in., nearly. Ans.

$$s = \frac{P}{k\,v^2} = \frac{p\,a}{k\,v^2}. \qquad \textbf{(39.)}$$

That is, the rubbing surface equals the total pressure divided by the coefficient of friction multiplied by the square of the velocity; or, it equals the pressure per square foot multiplied by the sectional area divided by the product of the coefficient of friction and the square of the velocity.

Example.—A gangway is 8′ × 8′; if the water-gauge shows ¾ inch and the velocity of the air is 280 feet per minute, what is the rubbing surface?

Solution.—The pressure per square foot is $p = 5.2\,W = 5.2 \times \frac{3}{4} = 3.9$ lb. per square foot; the sectional area is $8 \times 8 = 64$ sq. ft.

Hence, substituting in formula **39**,

$$s = \frac{p\,a}{k\,v^2};\ \text{or, } s = \frac{3.9 \times 64}{.0000000217 \times 280^2} = 146{,}713 \text{ sq. ft. Ans.}$$

$$v = \sqrt{\frac{p\,a}{k\,s}}. \qquad \textbf{(40.)}$$

That is, the velocity in feet per minute equals the square root of the pressure in pounds per square foot multiplied by the sectional area in square feet, divided by the product of the coefficient of friction and the rubbing surface in square feet.

Example.—In the last example suppose that the rubbing surface was known to be 146,713 square feet, and it was desired to find the velocity. Show how you would find it.

Solution.—Substituting the different values in formula **40**,

$$v = \sqrt{\frac{p\,a}{k\,s}} = \sqrt{\frac{3.9 \times 64}{.0000000217 \times 146{,}713}} = 280 \text{ ft. per min. Ans.}$$

When the total rubbing surface and the perimeter are known, the length of the airway may be found by means of the formula

$$l = \frac{s}{o}. \qquad \textbf{(41.)}$$

That is, the length of the airway is equal to the rubbing surface divided by the perimeter.

EXAMPLE.—The perimeter of an airway is 32 feet, and the rubbing surface is 146,713 feet; what is the length of the airway?

SOLUTION.—Applying formula **41**,

$$l = \frac{s}{o} = \frac{146,713}{32} = 4,585 \text{ ft. Ans.}$$

As before stated, the rubbing surface equals the product of the length and the perimeter; or,

$$s = l\,o. \qquad \textbf{(42.)}$$

THE QUANTITY OF AIR DISCHARGED.

963. Since a certain quantity of air is required to pass along the airway in order to secure the proper amount of ventilation, it is necessary to know how much air can be passed with a given velocity; or, knowing the quantity required, it is necessary to calculate the velocity, and from that to determine the pressure. If the velocity and sectional area are known, the quantity may be determined by the following formula, in which q = the quantity in cubic feet per minute:

$$q = a\,v. \qquad \textbf{(43.)}$$

That is, the quantity of air discharged in cubic feet per minute through a given airway is equal to the area of the section in square feet multiplied by the velocity in feet per minute.

964. A little consideration will show that formula **43** must be true; for, suppose that the sectional area is 1 square foot and the velocity is 1 foot per minute; then it is perfectly evident that the quantity discharged in 1 minute is 1 cubic foot. If the velocity be increased 2, 3, 4, etc., times, the number of cubic feet discharged will also be, respectively, 2, 3, 4, etc., times the original quantity; that is, the velocity will be 2, 3, 4, etc., feet per minute, and the quantity 2, 3, 4, etc., cubic feet per minute. Likewise, if the velocity remains at 1 cubic foot per minute, but with the area increased 2, 3, 4, etc., times, the quantity will be increased to 2, 3, 4, etc., cubic feet per minute. Consequently, if the area and velocity are both changed, the change in quantity must be the product of the two; that is, if the area be increased

from 1 square foot to, say, 26 square feet, and the velocity increased from 1 foot per minute to 1,000 feet per minute, the quantity will be increased from 1 cubic foot to $26 \times 1,000 = 26,000$ cubic feet per minute.

EXAMPLE.—A circular airway has a diameter of 9.026 feet, and the velocity of the air is 330 feet per minute; what is the quantity passing in cubic feet per minute?

SOLUTION.—Applying formula **43**, $a = 9.026^2 \times .7854 = 64$ sq. ft. Hence, $q = a\,v = 64 \times 330 = 21,120$ cu. ft. per minute. Ans.

965. If the quantity to be discharged and the sectional area are known, and it is required to find the velocity, use the following formula:

$$v = \frac{q}{a}. \qquad (44.)$$

That is, the velocity in feet per minute equals the quantity passing in cubic feet per minute divided by the sectional area in square feet.

EXAMPLE.—A circular airway has a diameter of 9.026 feet; what must be the velocity in order to pass 21,120 cubic feet per minute?

SOLUTION.—The sectional area was found to be 64 square feet in the last example. Hence, substituting in formula **44**,

$$v = \frac{q}{a} = \frac{21,120}{64} = 330 \text{ ft. per minute. Ans.}$$

966. The size of the airway usually depends upon other considerations than the quantity and velocity; but, in order to render the subject more complete, the following formula is given:

$$a = \frac{q}{v}. \qquad (45.)$$

That is, the sectional area equals the quantity in cubic feet per minute divided by the velocity in feet per minute.

967. Formulas **43**, **44**, and **45** may be combined with formulas **28** to **32**, so that the pressure (or velocity) may be determined at once, when the quantity and other needful data are known; but the simplest way is to calculate the velocity by formula **40**, and then substitute the value obtained in formula **43** to find the quantity; or to calculate

the velocity by formula **44,** and substitute in formula **37** or **38** to find the pressure. The formulas are given, however, in Table 24, and are there denoted by the letters **a, b, c,** etc., to distinguish them from the numbered formulas, which are considered to be more important.

EXAMPLE.—What is the total ventilating pressure required to pass 21,120 cubic feet of air per minute through an 8′ × 8′ air-course 6,000 feet long?

SOLUTION.—The sectional area $= 8 \times 8 = 64$ sq. ft. $= a$. The rubbing surface $= 8 \times 4 \times 6,000 = 192,000$ sq. ft. $= s$.

By formula **44,**

$$v = \frac{q}{a} = \frac{21,120}{64} = 330 \text{ ft. per minute.}$$

Therefore, applying formula **37,**

$$P = k\,s\,v^2 = .0000000217 \times 192,000 \times 330^2 = 453.72 \text{ lb. Ans.}$$

WORK AND POWER.

968. Work is equal to resistance in pounds multiplied by the space in feet through which the resistance is overcome. That is, suppose that it takes a force (pressure) of 25 pounds to move a certain body; then, if the resistance is uniform, as, for example, in lifting a weight, and the body is moved through a distance of 36 feet, the work done is $25 \times 36 = 900$ foot-pounds. Since time is not mentioned in the above definition, it follows that work is independent of the time; that is, no matter whether it takes 1 second or 1 year to move the body 36 feet, the work done is 900 foot-pounds.

Now, in order to compare the work done by different machines, time must be considered. Hence, the amount of work done in overcoming a resistance of 1 pound, through a space (distance) of 1 foot in 1 minute, is called the **unit of power.** The power of a machine is, then, the number of foot-pounds of work which it can perform in 1 minute, and this number divided by 33,000 is called the **horsepower of the machine.**

The power required to produce the proper ventilative effects may be easily calculated when the total pressure and

the velocity are known, or when the pressure per square foot and the quantity passed in cubic feet per minute are known. Thus, the total pressure represents the force required to overcome the resistances, and the velocity in feet per minute represents the space (distance) passed through in 1 minute; consequently, the product of the total pressure, P, and the velocity in feet per minute equals the work per minute, or the power. That is, representing the number of units of power by u,

$$u = Pv. \qquad \textbf{(46.)}$$

Likewise, since $P = pa$, $u = pav$; but, according to formula **43**, $av = q$; hence,

$$u = pq. \qquad \textbf{(47.)}$$

By dividing formulas **46** and **47** by 33,000, the horsepower may be found. Letting H represent the horsepower,

$$H = \frac{u}{33{,}000} = \frac{Pv}{33{,}000} = \frac{pav}{33{,}000} = \frac{pq}{33{,}000}. \qquad \textbf{(48.)}$$

EXAMPLE.—What horsepower is required to pass the air in the last example?

SOLUTION.—The total pressure was found to be 453.72 pounds, and the velocity 330 feet per minute. Hence, by formula **48**,

$$H = \frac{Pv}{33{,}000} = \frac{453.72 \times 330}{33{,}000} = 4.537 \text{ H. P., nearly. Ans.}$$

EXAMPLE.—If the water-gauge reading is 1.9 inches, and the quantity of air passing is 20,000 cubic feet per minute, what horsepower is required?

SOLUTION.—The pressure per square foot $= 5.2 \times 1.9 = 9.88$ lb. Therefore, applying formula **48**,

$$H = \frac{pq}{33{,}000} = \frac{9.88 \times 20{,}000}{33{,}000} = 6 \text{ H. P., nearly. Ans.}$$

969. From formula **48**, several other important formulas may be derived by a simple transposition of the terms. If the horsepower, sectional area, and the velocity of the air are known, and it is desired to find the ventilating pressure in pounds per square foot, the following formula may be used:

$$p = \frac{33{,}000\, H}{av}. \qquad \textbf{(49.)}$$

Or, if the horsepower and the quantity of air to be passed per minute are known, and p is required,

$$p = \frac{33,000\,H}{q}. \qquad \textbf{(50.)}$$

970. If it be required to ascertain the quantity which a certain horsepower will cause to pass with a given pressure, it may be found by formula **51**,

$$q = \frac{33,000\,H}{p}. \qquad \textbf{(51.)}$$

Similarly, the velocity may be found by formula **52**, when the horsepower and total pressure, or the horsepower, pressure per square foot, and sectional area are known.

$$v = \frac{33,000\,H}{P} = \frac{33,000\,H}{p\,a}. \qquad \textbf{(52.)}$$

EXAMPLE.—It is required to pass 20,000 cubic feet of air per minute. (*a*) What is the pressure per square foot if only 6 horsepower are required? (*b*) What is the water-gauge reading?

SOLUTION.—(*a*) Since only the horsepower and quantity are given, formula **50** must be used. Substituting,

$$p = \frac{33,000\,H}{q} = \frac{33,000 \times 6}{20,000} = 9.9 \text{ lb. per square foot. Ans.}$$

(*b*) Since $p = 5.2\,W$, $W = \frac{p}{5.2} = \frac{9.9}{5.2} = 1.9$ in., very nearly. Ans.

EXAMPLE.—Had the sectional area in the above example been 50 square feet, what would the velocity have been?

SOLUTION.—This example may be solved in two ways. By formula **44**,

$$v = \frac{q}{a} = \frac{20,000}{50} = 400 \text{ ft. per minute. Ans.}$$

By formula **52**,

$$v = \frac{33,000\,H}{p\,a} = \frac{38,000 \times 6}{9.9 \times 50} = 400 \text{ ft. per minute. Ans.}$$

971. Formulas **46** to **52** may be combined with formulas **37** to **42** to produce other formulas, which will shorten the work to some extent in certain cases; but the student will find it a better plan, as a rule, to calculate the pressure, velocity, or whatever he needs, by using one of

the formulas from **36** to **44**, and then substituting in one of the later formulas.

A number of these combination formulas will be given in Table 24, and the student may use them if he so chooses. One of these combination formulas is so important that it will now be given.

Multiplying both sides of formula **37** by v, $Pv = ksv^3$; but by formula **46**, $Pv = u$; hence,

$$u = ksv^3. \qquad \textbf{(53.)}$$

That is, the power in foot-pounds per minute equals the continued product of the coefficient of friction, the rubbing surface, and the **cube** *of the velocity.*

Likewise, since $u = pq$ (formula **47**), $pq = ksv^3$; or,

$$q = \frac{ksv^3}{p}. \qquad \textbf{(54.)}$$

EXAMPLE.—An air-course has a length of 4,752 feet; its perimeter is 30 feet, and its sectional area is 50 square feet. (*a*) What quantity of air will it pass at a velocity of 400 feet per minute? (*b*) What power will be required? The pressure is 9.9 pounds per square foot.

SOLUTION.—(*a*) This question is most easily solved by means of formula **43**, but to show the reliability of formula **54**, it will be solved both ways. By formula **43**,

$$q = av = 50 \times 400 = 20{,}000 \text{ cu. ft. per minute. Ans.}$$

By formula **54**, since $s = 4{,}752 \times 30 = 142{,}560$ sq. ft.

$$q = \frac{ksv^3}{p} = \frac{.0000000217 \times 142{,}560 \times 400^3}{9.9} = 20{,}000 \text{ cu. ft., nearly. Ans.}$$

(*b*) Substituting in formula **53**,

$$u = ksv^3 = .0000000217 \times 142{,}560 \times 400^3 =$$

$$198{,}000 \text{ ft.-lb. per minute, nearly}, = \frac{198{,}000}{33{,}000} = 6 \text{ H. P. Ans.}$$

972. There is one more combination formula which is chiefly valuable on account of the deductions which may be made from considerations of it, and which will now be given in order that the student may be able to answer a question sometimes asked at examinations for mine foremen's certificates. But, in order that an intelligent understanding may result, it is necessary to digress here and explain a certain geometrical law.

973. **Two figures are similar when the smaller may be so placed within the larger that their perimeters shall be parallel throughout their entire lengths, and their corresponding sides proportional.** Thus, if in Fig. 139 the perimeters of the two trapezoids are parallel when one is placed within the other, and $A\,D : a\,d :: B\,C : b\,c$, and the same relation is also true of any other two sides, as $A\,D : a\,d :: D\,C : d\,c$, then the two trapezoids are *similar*. Equilateral triangles, squares, and circles are always similar.

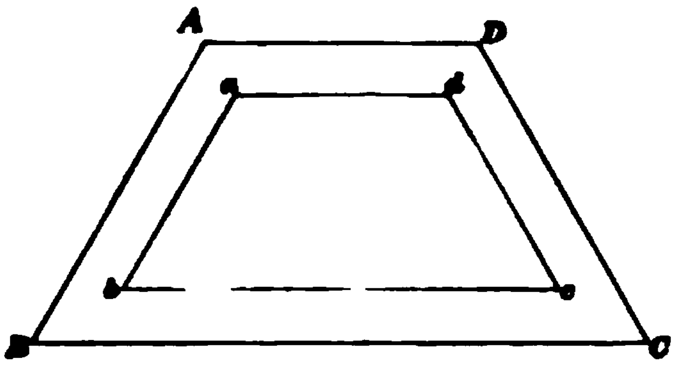


FIG. 139.

Now, it is proved in geometry that the areas of similar figures are to each other as the squares of any side, the squares of their perimeters, or the squares of any line similarly placed in them, as, for example, a diagonal or diameter. Also, that the volumes of similar solids are to each other as the cubes of similarly placed lines in them. Likewise, if *any* two similar figures are varied according to some power of similarly placed lines, all similar figures will vary according to the same powers of their similarly placed lines. For example, if the volume of a certain prism is 21 cubic inches and the length of a certain line in it is 2 inches, what will be the volume of a similar prism if the length of a similarly placed line in it is 3 inches? Since the volumes of similar solids are to each other as the cubes of their similar lines, $2^3 : 3^3 :: 21 : x$; or, $x = \frac{21 \times 27}{8} = 70.875$ cubic inches.

974. Returning now to the subject of ventilation, consider a mine having a square cross-section, and represent the length of a side by d. Then the area is d^2 and the perimeter is $4\,d$. According to formula **43**, $q = a\,v$; but, since for this case $a = d^2$, $q = d^2 v$. By formula **40**, $v = \sqrt{\frac{p\,a}{k\,s}} = \sqrt{\frac{p\,d^2}{k\,s}}$, since $a = d^2$. By formula **42**, $s = l\,o$, and since for this case $o = 4d$, $s = l \times 4d = 4ld$. Substituting

this value of s in the above formula for v, $v = \sqrt{\frac{p\,d^2}{k \times 4\,l\,d}} = \sqrt{\frac{p\,d}{4\,k\,l}}$. Substituting this value of v in the expression giving the value of q,

$$q = d^2 \times \sqrt{\frac{p\,d}{4\,k\,l}};\ \text{or, } q = \sqrt{\frac{p\,d^5}{4\,k\,l}}. \qquad \textbf{(55.)}$$

975. It must be remembered that formula **55** applies only to square airways. A consideration of it shows that if two square airways have the same length and pressure per square foot, the quantities of air which they will pass will be to each other as the square roots of the fifth powers of the lengths of their sides.

Also, if two square airways of different lengths are required to pass the same quantity of air with the same pressure per square foot, the lengths of the sides will be to each other as the fifth roots of the lengths of the airways.

Now, since squares are similar figures, the two statements just made will also apply to any two airways whose cross-sections are similar figures.

976. The following example is a question asked at an examination held at Pittsburg, in March, 1893:

Example.—If 15,000 cubic feet of air per minute are passing through an airway 4,000 feet in length, and 6 feet by 8 feet in section, what should be the dimensions of the section of another airway of precisely the same form (i. e., a similar section) to pass the same quantity of air, the length, however, being 3,000 feet, instead of 4,000 feet, as in the former case?

Solution.—Since the pressure is not stated, it is evidently intended to be the same in both cases. Then, according to the above statements, the lengths of similar sides are to each other as the fifth roots of the lengths. Hence, $6 : x :: \sqrt[5]{4{,}000} : \sqrt[5]{3{,}000}$;

or, $$x = 6 \times \frac{\sqrt[5]{3{,}000}}{\sqrt[5]{4{,}000}} = 6\sqrt[5]{\frac{3{,}000}{4{,}000}} = 6 \times .944 = 5.644 \text{ ft.}$$

Now, since the sections are similar, the sides are proportional; hence, $6 : 5.664 :: 8 : x$; or, $x = \frac{5.664 \times 8}{6} = 7.552$ ft. Therefore, the section is $5.664' \times 7.552'$. Ans.

977. From formula **55** there may also be deduced the proposition that if two square airways have the same length and pass the same quantities of air, the lengths of the sides of the airway will then vary *inversely* as the fifth roots of the pressures. This statement applies, of course, to airways of similar sections. Conversely, the pressures vary inversely as the fifth powers of the lengths of the sides.

A method of finding the fifth roots of numbers will be given hereafter. (See Art. **1000.**)

978. Other Resistances.—All that is necessary for the calculation of the resistance of the flow of air through a straight airway has now been given, and all that remains to be considered, so far as appertains to the flow of air, are those effects produced by bends, contractions, or enlargements of the sections, and splits. Each of the foregoing results in a change in the velocity of the flowing air, and, consequently, in a change in the ventilating pressure. The losses due to bends are considerable, particularly a bend of 90° or greater. Where a bend is absolutely necessary, the corners should be rounded (if practicable) to as large a radius as possible, if it is desired to reduce the mine resistance to a minimum. There is no reliable formula for calculating the resistance due to bends, but they certainly reduce the velocity to a great extent, especially a bend of 90° or greater. If the reduction or enlargement of the sectional area is slight compared with the sectional area of the airway, the consequent loss of velocity may be disregarded entirely. In any case, it is a difficult matter to decide how much to allow for such loss. The losses due to regulators and to splits will be treated separately in a later section. Since, as before mentioned, the coefficient of friction, .0000000217, is very high, much above what would actually be obtained in practice for a straight airway, the losses due to bends, enlargements, and contractions may be neglected altogether without any material error, the airway being calculated as if it were straight and of uniform section through-

out. This statement applies, of course, to slight enlargements or contractions.

979. Formulas.—Formulas **36** to **54**, inclusive, and others which are not quite as important, are given in Table 24, so as to be convenient for reference. The formulas not previously given are denoted by the letters **a, b, c,** etc. A specimen calculation is also worked out with each formula. To prevent any misconception, the letters and their meanings are repeated below:

a = sectional area of airway in square feet;
H = horsepower;
k = coefficient of friction = .0000000217;
l = length of airway in feet;
o = perimeter of airway in feet;
p = ventilating pressure in pounds per square foot;
P = total ventilating pressure in pounds;
q = quantity of air in cubic feet per minute;
s = rubbing surface in square feet;
u = units of power in foot-pounds per minute;
v = velocity in feet per minute;
W = water-gauge in inches of water.

To render the formulas more convenient for reference, they are not given in sequence according to their numbers, but are classified according to the letters whose values it is desired to find, the letters having the meaning given above.

The basis for the calculations is an airway 5 feet wide by 4 feet high and 2,000 feet long, the velocity to be 500 feet per minute.

TABLE 24.

Formulas.		Specimen Calculations.
To find the area:		
$a = \frac{P}{p}$.	(a.)	$a = \frac{195.3}{9.765} = 20$ sq. ft. Ans.
$a = \frac{k s v^2}{p}$.	(a'.)	$a = \frac{.0000000217 \times 36,000 \times 500^2}{9.765} = 20$ sq. ft. Ans.
$a = \frac{q}{v}$.	(45.)	$a = \frac{10,000}{500} = 20$ sq. ft. Ans.
$a = \frac{u}{p v}$.	(b.)	$a = \frac{97,650}{9.765 \times 500} = 20$ sq. ft. Ans.
$a = \frac{33,000 H}{p v}$.	(c.)	$a = \frac{33,000 \times 2.959}{9.765 \times 500} = 20$ sq. ft. Ans.
$a = \frac{k s v^2 q}{u}$.	(d.)	$a = \frac{.0000000217 \times 36,000 \times 500^2 \times 10,000}{97,650} = 20$ sq. ft. Ans.
To find the horsepower:		
$H = \frac{u}{33,000}$.	(48.)	$H = \frac{97,650}{33,000} = 2.959$ horsepower. Ans.
$H = \frac{P v}{33,000}$.	(48.)	$H = \frac{195.3 \times 500}{33,000} = 2.959$ horsepower. Ans.
$H = \frac{p q}{33,000}$.	(48.)	$H = \frac{9.765 \times 10,000}{33,000} = 2.959$ horsepower. Ans.

TABLE 24—*Continued.*

Formulas.		Specimen Calculations.
$H = \frac{p\,a\,v}{33,000}$.	(48.)	$H = \frac{9.765 \times 20 \times 500}{33,000} = 2.959$ horsepower. Ans.
To find the coefficient of friction:		
$k = \frac{P}{s\,v^2}$.	(e.)	$k = \frac{195.3}{36,000 \times 500^2} = .0000000217$ lb. per sq. ft. per minute. Ans.
$k = \frac{p\,a}{s\,v^3}$.	(f.)	$k = \frac{9.765 \times 20}{36,000 \times 500^2} = .0000000217$ lb. per sq. ft. per minute. Ans.
$k = \frac{u}{s\,v^3}$.	(g.)	$k = \frac{97,650}{36,000 \times 500^3} = .0000000217$ lb. per sq. ft. per minute. Ans.
$k = \frac{p\,q}{s\,v^3}$.	(h.)	$k = \frac{9.765 \times 10,000}{36,000 \times 500^3} = .0000000217$ lb. per sq. ft. per minute. Ans.
To find the length of the airway:		
$l = \frac{s}{o}$.	(41.)	$l = \frac{36,000}{18} = 2,000$ ft. Ans.
To find the perimeter of the airway:		
$o = \frac{s}{l}$.	(i.)	$o = \frac{36,000}{2,000} = 18$ ft. Ans.

TABLE 24—*Continued.*

Formulas.		Specimen Calculations.
To find the total pressure:		
$P = p\,a.$	**(36.)**	$P = 9.765 \times 20 = 195.8$ lb. Ans.
$P = k\,s\,v^2.$	**(37.)**	$P = .0000000217 \times 36{,}000 \times 500^2 = 195.8$ lb. Ans.
$P = \frac{u}{v}.$	**(j.)**	$P = \frac{97{,}650}{500} = 195.3$ lb. Ans.
$P = \frac{33{,}000\,H}{v}.$	**(k.)**	$P = \frac{33{,}000 \times 2.959}{500} = 195.3$ lb. Ans.
$P = \frac{k\,s\,q^2}{a^2}.$	**(l.)**	$P = \frac{.0000000217 \times 36{,}000 \times 10{,}000^2}{20^2} = 195.3$ lb. Ans.
To find the pressure in pounds per square foot:		
$p = \frac{P}{a}.$	**(m.)**	$p = \frac{195.3}{20} = 9.765$ lb. per square foot. Ans.
$p = \frac{k\,s\,v^2}{a}.$	**(38.)**	$p = \frac{.0000000217 \times 36{,}000 \times 500^2}{20} = 9.765$ lb. per square foot. Ans.
$p = \frac{u}{q}.$	**(n.)**	$p = \frac{97{,}650}{10{,}000} = 9.765$ lb. per square foot. Ans.
$p = \frac{33{,}000\,H}{a\,v}.$	**(49.)**	$p = \frac{33{,}000 \times 2.959}{20 \times 500} = 9.765$ lb. per square foot. Ans.
$p = \frac{33{,}000\,H}{q}.$	**(50.)**	$p = \frac{33{,}000 \times 2.959}{10{,}000} = 9.765$ lb. per square foot. Ans.

TABLE 24—*Continued.*

Formulas.		Specimen Calculations.
$p = \frac{k s v^3}{q}$.	(o.)	$p = \frac{.0000000217 \times 36.000 \times 500^3}{10.000} = 9.765$ lb. per square foot. Ans.
$p = \frac{k s q^3}{a^3}$.	(o′.)	$p = \frac{.0000000217 \times 36.000 \times 10,000^3}{20^3} = 9.765$ lb. per square foot. Ans.
$p = 5.2\,W$.		$p = 5.2 \times 1.87788 = 9.765$ lb. per square foot. Ans.
To find the quantity of air passing in cubic feet per minute:		
$q = a\,v$.	(43.)	$q = 20 \times 500 = 10,000$ cu. ft. per minute. Ans.
$q = \frac{u}{p}$.	(p.)	$q = \frac{97,650}{9.765} = 10,000$ cu. ft. per minute. Ans.
$q = \frac{33,000\,H}{p}$.	(51.)	$q = \frac{33,000 \times 2.959}{9.765} = 10,000$ cu. ft. per minute. Ans.
$q = \frac{k s v^3}{p}$.	(54.)	$q = \frac{.0000000217 \times 36,000 \times 500^3}{9.765} = 10,000$ cu. ft. per minute. Ans.
$q = a\sqrt{\frac{p\,a}{k\,s}}$.	(q.)	$q = 20\sqrt{\frac{9.765 \times 20}{.0000000217 \times 36,000}} = 10,000$ cu. ft. per minute. Ans.
To find the rubbing surface in square feet:		
$s = \frac{P}{k\,v^2}$.	(39.)	$s = \frac{195.3}{.0000000217 \times 500^2} = 36\,000$ sq. ft. Ans.
$s = \frac{p\,a}{k\,v^2}$.	(39.)	$s = \frac{9.765 \times 20}{.0000000217 \times 500^2} = 36.000$ sq. ft. Ans.

TABLE 24—*Continued.*

Formulas.		Specimen Calculations.
$s = l\,o.$	**(42.)**	$s = 2{,}000 \times 18 = 36.000$ sq. ft. Ans.
$s = \dfrac{u}{k\,v^3}.$	**(r.)**	$s = \dfrac{97\ 650}{.0000000217 \times 500^3} = 36\ 000$ sq. ft. Ans.
$s = \dfrac{p\,q}{k\,v^3}.$	**(s.)**	$s = \dfrac{9.765 \times 10.000}{.0000000217 \times 500^3} = 36{,}000$ sq. ft. Ans.
To find the units of power in foot-pounds per minute:		
$u = P\,v.$	**(46.)**	$u = 195.3 \times 500 = 97.650$ ft.-lb. per minute. Ans.
$u = p\,q.$	**(47.)**	$u = 9.765 \times 10.000 = 97{,}650$ ft.-lb. per minute. Ans.
$u = 33{,}000\,H.$	**(t.)**	$u = 33.000 \times 2.959 = 97{,}650$ ft.-lb. per minute. Ans.
$u = p\,a\,v.$	**(u.)**	$u = 9.765 \times 20 \times 500 = 97{,}650$ ft.-lb. per minute. Ans.
$u = k\,s\,v^3.$	**(53.)**	$u = .0000000217 \times 36{,}000 \times 500^3 = 97{,}650$ ft.-lb. per minute. Ans.
$u = \dfrac{k\,s\,q^3}{a^3}$	**(u'.)**	$u = \dfrac{.0000000217 \times 36{,}000 \times 10.000^3}{20^3} = 97{,}650$ ft.-lb. per minute. Ans.
To find the velocity in feet per minute:		
$v = \sqrt{\dfrac{p\,a}{k\,s}}.$	**(40.)**	$v = \sqrt{\dfrac{9.765 \times 20}{.0000000217 \times 36{,}000}} = 500$ ft. per minute. Ans.
$v = \sqrt{\dfrac{P}{k\,s}}.$	**(v.)**	$v = \sqrt{\dfrac{195.3}{.0000000217 \times 36{,}000}} = 500$ ft. per minute. Ans.

TABLE 24—*Concluded.*

Formulas.		Specimen Calculations.
$v = \frac{q}{a}$.	(44.)	$v = \frac{10,000}{20} = 500$ ft. per minute. Ans.
$v = \frac{u}{p}$.	(w.)	$v = \frac{97,650}{195.3} = 500$ ft. per minute. Ans.
$v = \frac{u}{p\,a}$.	(x.)	$v = \frac{97,650}{9.765 \times 20} = 500$ ft. per minute. Ans.
$v = \frac{33,000\,H}{p}$.	(y.)	$v = \frac{33,000 \times 2.959}{195.3} = 500$ ft. per minute. Ans.
$v = \frac{33,000\,H}{p\,a}$.	(y'.)	$v = \frac{33,000 \times 2.959}{9.765 \times 20} = 500$ ft. per minute. Ans.
$v = \sqrt[3]{\frac{u}{k\,s}}$.	(z.)	$v =$ $= 500$ ft. per minute. Ans.
$v = \sqrt[3]{\frac{p\,q}{k\,s}}$.	(z'.)	$v =$ $= 500$ ft. per minute. Ans.
To find the water-gauge:		
$W = \frac{p}{5.2}$.		$W = \frac{9.765}{5.2} = 1.87788$ in. Ans.

NOTE.—The water-gauge is calculated to five decimal places, so that it will correspond to the other values; two places are sufficient in practice.

LAWS OF VENTILATION.

980. In order to ascertain the effects produced by varying the airway or by varying the quantity, velocity, etc., of the air, it is generally easier to make use of one of the following *laws* than to solve by means of one of the foregoing formulas. The laws are also useful for comparing the results obtained from two airways. Letting p, q, v, s, etc., represent, respectively, the pressure, quantity, velocity, rubbing surface, etc., before the change, and p_1, q_1, v_1, s_1, etc., the same things after the change, the laws may be stated as follows:

(1) The pressure varies directly as the extent of the rubbing surface; i. e., $p : p_1 :: s : s_1$, or $P : P_1 :: s : s_1$.

(2) The pressure varies directly as the density* of the air; i. e., $p : p_1 :: w : w_1$, or $P : P_1 :: w : w_1$.

(3) The pressure varies directly as the square of the quantity; i. e., $p : p_1 :: q^2 : q_1^2$, or $P : P_1 :: q^2 : q_1^2$.

(4) The pressure varies directly as the square of the velocity; i. e., $p : p_1 :: v^2 : v_1^2$, or $P : P_1 :: v^2 : v_1^2$.

(5) The pressure varies directly as the length of the airway; i. e., $p : p_1 :: l : l_1$, or $P : P_1 :: l : l_1$.

(6) The pressure varies directly as the length of the perimeter; i. e., $p : p_1 :: o : o_1$, or $P : P_1 :: o : o_1$.

(7) The pressure per square foot varies inversely as the area of the airway; i. e., $p : p_1 :: a_1 : a$.

(8) The quantity varies directly as the square root of the pressure; i. e., $q : q_1 :: \sqrt{p} : \sqrt{p_1}$, or $q : q_1 :: \sqrt{P} : \sqrt{P_1}$.

(9) The quantity varies directly as the cube root of the power; i. e., $q : q_1 :: \sqrt[3]{u} : \sqrt[3]{u_1}$, or $q : q_1 :: \sqrt[3]{H} : \sqrt[3]{H_1}$.

(10) The quantity varies inversely as the square root of the rubbing surface; i. e., $q : q_1 :: \sqrt{s_1} : \sqrt{s}$.

(11) The velocity varies directly as the square root of the pressure; i. e., $v : v_1 :: \sqrt{p} : \sqrt{p_1}$, or $v : v_1 :: \sqrt{P} : \sqrt{P_1}$.

---

* By density is meant the *weight of a cubic foot in pounds.*

(12) The velocity varies directly as the square root of the area; i. e., $v : v_1 :: \sqrt{a} : \sqrt{a_1}$.

(13) The velocity varies inversely as the square root of the length of the airway; i. e., $v : v_1 :: \sqrt{l_1} : \sqrt{l}$.

(14) The velocity varies inversely as the square root of the rubbing surface; i. e., $v : v_1 :: \sqrt{s_1} : \sqrt{s}$.

(15) The power varies directly as the cube of the quantity; i. e., $u : u_1 :: q^3 : q_1^3$, or $H : H_1 :: q^3 : q_1^3$.

(16) The rubbing surface varies inversely as the square of the quantity; i. e., $s : s_1 :: q_1^2 : q^2$.

(17) The rubbing surface varies inversely as the square of the velocity; i. e., $s : s_1 :: v_1^2 : v^2$.

(18) The sectional area varies directly as the square of the velocity; i. e., $a : a_1 :: v^2 : v_1^2$.

(19) The length of the airway varies inversely as the square of the velocity; i. e., $l : l_1 :: v_1^2 : v^2$.

(20) The length of the airway varies inversely as the square of the quantity; i. e., $l : l_1 :: q_1^2 : q^2$.

For similar airways, let d equal the length of a side; then,

(21) The quantity varies directly as the square root of the fifth power of the length of the side; i. e., $q : q_1 :: \sqrt{d^5} : \sqrt{d_1^5}$.

(22) The pressure varies inversely as the fifth power of the length of the side; i. e., $p : p_1 :: d_1^5 : d^5$.

(23) The length of the side varies inversely as the fifth root of the pressure; i. e., $d : d_1 :: \sqrt[5]{p_1} : \sqrt[5]{p}$.

(24) The length of the side varies directly as the fifth root of the square of the quantity; i. e., $d : d_1 :: \sqrt[5]{q^2} : \sqrt[5]{q_1^2}$.

To the above laws may also be added another:

(25) If equal quantities of air pass through two airways, the velocities will vary inversely as the areas; i. e., $v : v_1 :: a_1 : a$.

PRACTICAL PROBLEMS.

981. To illustrate the application of the foregoing laws and formulas, a series of practical examples such as are asked at examinations for mine foremen, together with their solutions, will now be given. By paying particular and careful attention to the statements of the examples and the solutions following them, the student should then be able to work similar ones without trouble. The above twenty-five laws should be carefully memorized, so that the student will not be obliged to refer to them.

1. What quantity of air is passing down a shaft 12 feet in diameter when the current has a velocity of 325 feet per minute?

SOLUTION.—Since the diameter is specified, the shaft is evidently circular. Applying formula **43**,

$$q = a\,v = 12^2 \times .7854 \times 325 = 36{,}756.72 \text{ cu. ft. per minute.} \quad \text{Ans.}$$

2. Where the airway is 12 feet wide at the bottom, 10 ft. 4 in. wide at the top, and 6 ft. 6 in. high, and the velocity of the air is 340 feet per minute, what is (*a*) the sectional area of the airway, and (*b*) the quantity of air passing per minute?

SOLUTION.—(*a*) The section is a trapezoid; hence,

$$\text{area} = \frac{10\tfrac{1}{3} + 12}{2} \times 6\tfrac{1}{2} = 72\tfrac{7}{12} \text{ sq. ft.,}$$

since 4 in. $= \tfrac{1}{3}$ ft., and 6 in. $= \tfrac{1}{2}$ ft. Ans.

(*b*) Applying formula **43**,

$$q = a\,v = 72\tfrac{7}{12} \times 340 = 24{,}678\tfrac{1}{3} \text{ cu. ft. per minute.} \quad \text{Ans.}$$

3. If a shaft 8 ft. by 24 ft. in section is the intake, and the fan is exhausting 160,000 cubic feet of air per minute, what is the velocity of the air-current in the shaft?

SOLUTION.—Applying formula **44**,

$$v = \frac{q}{a} = \frac{160{,}000}{8 \times 24} = 833\tfrac{1}{3} \text{ ft. per minute.} \quad \text{Ans.}$$

4. The section of an airway is a right-angled triangle, 10 feet wide at the base and 7½ feet high; what quantity of air is passing when the velocity is 280 feet per minute?

SOLUTION.—Area of section $= \dfrac{10 \times 7.5}{2} = 37.5$ sq. ft. Then, applying formula **43**,

$$q = a\,v = 37.5 \times 280 = 10{,}500 \text{ cu. ft. per minute.} \quad \text{Ans.}$$

5. An air-course is 500 yards long, 6 feet high, and 7 feet wide; what is (*a*) its sectional area, (*b*) its perimeter, and (*c*) its rubbing surface?

SOLUTION.—(*a*) Sectional area $a = 6 \times 7 = 42$ sq. ft. Ans.

(*b*) Perimeter $o = 6 \times 2 + 7 \times 2 = 26$ ft. Ans.

(*c*) Applying formula **42**,

$$s = l\,o = 500 \times 3 \times 26 = 39{,}000 \text{ sq. ft.} \quad \text{Ans.}$$

6. The rubbing surface is 25,000 sq. ft. and the perimeter 50 ft.; what is the length?

SOLUTION.—Applying formula **41**,

$$l = \frac{s}{o} = \frac{25{,}000}{50} = 500 \text{ ft} \quad \text{Ans.}$$

7. When the water-gauge is 1.85 in., what pressure per square foot does it indicate?

SOLUTION.— $p = 5.2\,W = 5.2 \times 1.85 = 9.62$ lb. per square foot. Ans.

8. What is the total ventilating pressure of an airway 6 feet by 7 feet, the water-gauge being .5 of an inch?

SOLUTION.—Pressure per square foot $= 5.2 \times .5 = 2.6$ lb.; area $= 6 \times 7 = 42$ sq. ft. Applying formula **36**,

$$P = p\,a = 2.6 \times 42 = 109.2 \text{ lb.} \quad \text{Ans.}$$

9. What quantity of air is passing through an airway 7 feet high by 7 feet wide when the velocity of the current is 300 feet per minute?

SOLUTION.—Applying formula **43**,

$$q = a\,v = 7 \times 7 \times 300 = 14{,}700 \text{ cu. ft. per minute.} \quad \text{Ans.}$$

10. If 80,000 cubic feet of air are required per minute in a mine, and the shaft velocity must not exceed 800 feet per minute, what is the smallest sectional area that the shaft may have?

SOLUTION.—Using formula **45**,

$$a = \frac{q}{v} = \frac{80{,}000}{800} = 100 \text{ sq. ft.} \quad \text{Ans.}$$

11. Suppose a gangway 10 feet by 10 feet and 1,000 feet long, in which the air has a velocity of 450 feet per minute, and the pressure as indicated by the water-gauge is 2 pounds; what is (*a*) the water-gauge reading, (*b*) the quantity of air passing per minute, and (*c*) the horsepower?

SOLUTION.—(*a*) Water-gauge $= W = \dfrac{2}{5.2} = .38$ in. Ans.

(*b*) Applying formula **43**,

$$q = a\,v = 10 \times 10 \times 450 = 45{,}000 \text{ cu. ft. per minute.} \quad \text{Ans.}$$

(*c*) Using formula **48**,

$$H = \frac{p\,q}{33{,}000} = \frac{2 \times 45{,}000}{33{,}000} = 2.727 \text{ H. P.} \quad \text{Ans.}$$

12. If you have two airways under the same pressure, one 6 feet wide, 6 feet high, and 5,000 feet long, the other 8 feet wide, $4\frac{1}{2}$ feet high, and 5,000 feet long, which will pass the greater quantity of air, and why?

SOLUTION.—Since the pressure and length remain the same, it is evident that the airway having the smaller perimeter will pass the greater quantity, since the rubbing surface will be less; perimeter of first airway $= 6 \times 4 = 24$ ft.; of the second airway, $8 \times 2 + 4\frac{1}{2} \times 2 = 25$ ft. Representing by 1 the amount passed by the first airway, and applying law (**10**), $1 : q_1 :: \sqrt{25} : \sqrt{24}$, or $q_1 = .98$; i. e., the second airway will pass 98% of the amount passed by the first airway. Ans.

13. The pressure producing ventilation is 7.8 pounds per square foot; what is the water-gauge?

SOLUTION.— $W = \frac{p}{5.2} = \frac{7.8}{5.2} = 1.5 = 1\frac{1}{2}$ in. Ans.

14. When the quantity of air passing is 60,000 cubic feet, with a water-gauge of 1.5 inches, what are the units of power producing ventilation?

SOLUTION.—Pressure $= p = 5.2\,W = 5.2 \times 1.5 = 7.8$ lb. per square foot. Using formula **47**,

$$u = p\,q = 7.8 \times 60{,}000 = 468{,}000 \text{ ft. lb. per minute.} \quad \text{Ans.}$$

15. How many horsepower are represented by 468,000 units of power?

SOLUTION.—Using formula **48**,

$$H = \frac{u}{33{,}000} = \frac{468{,}000}{33{,}000} = 14.18 \text{ H. P., nearly.} \quad \text{Ans.}$$

16. With a water-gauge of $\frac{6}{10}$ of an inch, the quantity of air passing is 24,000 cubic feet per minute; what water-gauge will be required to pass 36,000 cubic feet per minute?

SOLUTION.—Since the water-gauge and pressure are directly proportional to each other, law (**3**) may be applied; or,

$$24{,}000^2 : 36{,}000^2 :: .6 : x; \text{ whence, } x = 1.35 \text{ in. of water.} \quad \text{Ans.}$$

17. If 16,500 cubic feet of air are passing per minute with a pressure of 4.68 pounds per square foot, what quantity will pass with a pressure of 6.24 pounds per square foot?

SOLUTION.—Applying law (**8**),

$$16{,}500 : q_1 :: \sqrt{4.68} : \sqrt{6.24}; \text{ or, } q_1 = 19{,}052 \text{ cu. ft. per minute.} \quad \text{Ans.}$$

18. If 3 horsepower pass 15,000 cubic feet of air per minute, what horsepower would be required to double the quantity?

SOLUTION.—Applying law (**15**),

$$3 : H_1 :: 15{,}000^3 : (15{,}000 \times 2)^3; \text{ or, } H_1 = 24 \text{ H. P. Ans.}$$

19. Is there any disadvantage or loss in having the air travel at a high speed?

SOLUTION.—There is a very decided loss; for, according to the third law of friction, the pressure varies as the square of the velocity. If, therefore, the velocity is to be doubled, the pressure must be increased as the *square of two;* that is, four times. If the velocity is to be trebled, the pressure must be increased as the square of three; that is, nine times.

20. If 32,000 cubic feet of air are passing through an airway 6′ × 5′, under a pressure of 3.6 pounds per square foot, what pressure is necessary in an airway 9′ × 5′ to pass the same quantity?

SOLUTION.—Call the resistance of the first airway A, and that of the second one B, and call the required pressure x; then, $A : B :: 3.6 : x$, or $x = \frac{B}{A} \times 3.6$, because the pressures vary directly as the resistances.

$$x = \frac{\left(\frac{32{,}000}{45}\right)^2 \times \frac{28}{45} \times 3.6}{\left(\frac{32{,}000}{30}\right)^2 \times \frac{22}{30}}; \text{ or, by cancelation, } x = \frac{\frac{28}{45^3} \times 3.6}{\frac{22}{30^3}}.$$

Further, $x = \frac{30^3}{22} \times \frac{28}{45^3} \times 3.6 = \frac{2^3}{11} \times \frac{14}{3^3} \times 3.6 = 1.3575$ lb., the required pressure per square foot. Ans.

21. If a pressure of 3.2 pounds per square foot produces a velocity of 560 feet per minute, what pressure is required to produce a velocity of 700 feet per minute in the same airway?

SOLUTION.—Applying law (**4**),

$$3.2 : p_1 :: 560^2 : 700^2; \text{ whence, } p_1 = 5 \text{ lb. per square foot. Ans.}$$

22. If 24,000 cubic feet are passing through an airway having a rubbing surface of 75,000 square feet, what quantity will pass if the rubbing surface is increased to 100,000 square feet, the increase of rubbing surface being due to the lengthening of the airway?

SOLUTION.—Applying law (**10**),

$$24{,}000 : q_1 :: \sqrt{100{,}000} : \sqrt{75{,}000}; \text{ or, } q_1 = 20{,}785 \text{ cu. ft. Ans.}$$

23. If in an airway 1,200 feet long the air has a velocity of 400 feet per minute under a pressure of 3 pounds per square foot, what must the pressure be to maintain the same velocity if the length of airway is increased to 1,800 feet?

SOLUTION.—Applying law (**5**),

$3 : p_1 :: 1{,}200 : 1{,}800$; or, $p_1 = 4.5$ lb. per square foot. Ans.

24. If the air passes with a velocity of 600 feet per minute through an airway whose sectional area is 64 square feet, what will the velocity be if the area is decreased to 48 square feet, the pressure remaining constant?

SOLUTION.—Applying law (**12**),

$600 : v_1 :: \sqrt{64} : \sqrt{48}$; or, $v_1 = 519.6$ ft. per minute. Ans.

25. Two circular airways of the same length have diameters of 3 feet and 4 feet, respectively; if a pressure of 5 pounds per square foot will force the air through the 4-foot airway, what pressure is required to pass the same quantity through the 3-foot airway?

SOLUTION.—Law (**22**) must be applied to this case, and since a circle has no sides, the perimeter or diameter may be used in the proportion. Hence, $5 : p_1 :: 3^5 : 4^5$; or, $5 : p_1 :: 243 : 1{,}024$; whence, $p_1 = 21.07$ lb. per square foot. Ans.

26. If 10,000 cubic feet of air pass per minute through a circular airway 12 feet in diameter, how many cubic feet per minute will pass through an airway 6 feet in diameter and having the same length, the pressure being the same in both cases?

SOLUTION.—Applying law (**21**),

$10{,}000 : q_1 :: \sqrt{12^5} : \sqrt{6^5}$; or, $q_1 = 1{,}768$ cu. ft. per minute. Ans.

982. Remarks.—By aid of the foregoing laws and formulas, the student can calculate any problem relating to the flow of air which does not involve splits or regulators. Many of the formulas would be unnecessary if the student had even a slight knowledge of algebra. For example, formulas **a′**, **f**, **39** and **40** are all derived from formula **38** by simply transposing terms; and formula **38** is, in turn, derived from formula **37** by substituting for P its value $p\ a$. If the student has no knowledge of algebra, he should memorize all of the numbered formulas.

983. In working examples, the student should proceed as follows: Consider example 11, Art. **981.** First ascertain what is required. In this example we want the water-gauge, the quantity, and the horsepower. The water-gauge is easily obtained, since p, the pressure per square foot, is given. To find q, the quantity, we look in Table 24, and find that there are five formulas by which the value of q may be obtained.

We can not use **p** or **51**, because we do not know the values of u or H; but we can use any one of the three remaining formulas, for we know, or can readily find, the values of a, v, s, and k, which are given in the example. Such being the case, we naturally use the one which will require the least amount of labor, and that is, evidently, formula **43.** To find H, the horsepower, we refer again to Table 24, and find four different forms of formula **48,** any one of which may be used, since u, in the first form, may be found by means of formula **47.** We again use the easiest one, which is the one used in the solution, since the values of p and q are both known.

Examples solved like Example 18, Art. **981,** are worked in a similar manner. First find what is wanted (in this case H_1) and what is given (in this case H, q, and q_1); then look in the list of laws (Art. **980**) for one giving the relation between the horsepower and the quantity (in this case law **15**).

INFLUENCE OF A STACK UPON THE MOTIVE COLUMN.

984. The erection of a stack over the upcast shaft has the same effect as increasing the effective depth of the shaft, or, in other words, increasing the height of the motive column. The quantity of air in circulation is thereby increased according to the proportion

$$\sqrt{D} : \sqrt{D+h} :: q_1 : q_2;$$

or,

$$q_2 = q_1 \sqrt{\frac{D+h}{D}},$$

and

$$q_1 = q_2 \sqrt{\frac{D}{D+h}}, \qquad (55_1.)$$

in which q_1 = quantity of air in circulation per minute without the stack;

q_2 = quantity of air in circulation per minute after the stack is erected;

D = depth of shaft;

h = height of stack.

Example.—The depth of a certain furnace shaft is 225 feet, the height of the stack over it is 31 feet; to what will a circulation of 24,000 cu. ft. per minute be reduced if the stack is blown down?

Solution.—Applying formula **55₁**,

$$q_1 = 24{,}000\sqrt{\frac{225}{225+31}} = 24{,}000 \times \frac{15}{16} = 22{,}500 \text{ cu. ft. per minute. Ans.}$$

SPLITTING THE AIR.

985. By splitting is here meant dividing the ventilating current into two or more currents, each of which circulates in a separate district of the mine; any currents thus formed is commonly called a **split.**

The benefits to be derived from the splitting of the air-current may be stated as follows:

(*a*) A larger volume of air may be circulated in a mine with the same power.

(*b*) Fresher and purer air is supplied at the working face in each district, and the velocity of the current traversing the face is moderate.

(*c*) Each district has its own circulation, which is readily controlled, and may be increased or decreased as occasion may require.

(*d*) An explosion or a windy shot occurring in one district is not as often transmitted throughout the mine.

The student will realize at once the importance of a thorough knowledge of this portion of the subject. For example, the means employed for the ventilation of a new mine, whether fan or furnace, are often found after a year or two of rapid development to be inadequate to the present need. This difficulty is easily overcome in most cases by judicious splitting of the ventilating current.

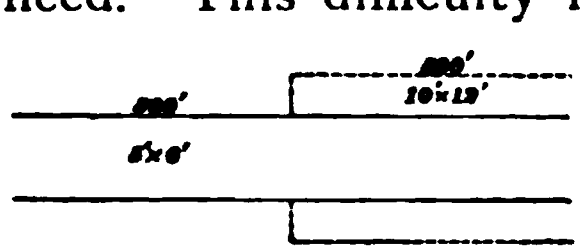


Fig. 140.

986. In order to study the effects of splitting, consider Fig. 140, which represents an airway 1,000 feet long and 5′ × 6′ in section. Suppose that 12,000 cubic feet per minute are

passing; then, the velocity is $\frac{12,000}{5 \times 6} = 400$ feet per minute. The rubbing surface is $(2 \times 5 + 2 \times 6) \times 1,000 = 22,000$ square feet. Hence, by formula **53**, $u = k s v^2 = .0000000217 \times 22,000 \times 400^2 = 30,553.6$ foot-pounds per minute.

Suppose that midway in its length the airway were to be enlarged, as shown by the dotted lines. The velocity in the large airway must now be greatly reduced, since the quantity discharged is the same as before (assuming that the velocity in the small airway remains 400 feet per minute), and the area of the section being larger, the velocity must be less. By formula **44**, the velocity $= v = \frac{q}{a} = \frac{12,000}{10 \times 12} = 100$ feet per minute. Now, since the rubbing surface of the small airway is just one-half of what it was before, the power required to force the air through it must evidently be one-half, or $\frac{30,553.6}{2} = 15,276.8$ foot-pounds per minute. The power required to force the air through the large airway is (see formula **47**) $u = p\,q = \frac{k s v^2}{a} \times q =$

$$\frac{.0000000217 \times [(2 \times 10 + 2 \times 12) \times 500] \times 100^2}{10 \times 12} \times 12,000 =$$

477.4 ft.-lb. per minute. Hence, the total power $= 15,276.8 + 477.4 = 15,754.2$ foot-pounds per minute, while in the former case, 30,553.6 foot-pounds per minute were required. This shows that by enlarging the airway, as shown, the same quantity may be passed with a greatly reduced power, or the quantity may be greatly increased with the same power. The quantity that will pass with the same power is easily found by applying law **15**. Thus,

$$15,754.2 : 30,553.6 :: 12,000^3 : q_1^3,$$

or $$q_1 = 14,965 \text{ cu. ft. per min.}$$

Since, before the enlargement, 12,000 cubic feet were discharged, the gain is $14,965 - 12,000 = 2,965$ cubic feet per minute, or very nearly 25 per cent.

The above calculation shows that in the case just considered the small airway requires $\frac{15,276.8}{477.4} = 32$ times as much power as the large one, and that if the small airway be decreased in length, this proportion of 32 to 1 will also decrease. Consequently, in cases like the above, it is best to decrease the length of the small airway as much as possible.

986₁. To split the air-current to advantage requires that the *main* airways, both *intake* and *return*, as also the *downcast* and *upcast* shafts, shall be of ample sectional area. A wise provision should always be made in this respect, since the entire circulation for all the districts must pass through these main airways. Were it not for this crowding of all the circulation into these main conduits for a short distance, the volume of air produced, when the power remains constant, would be always proportional to the number of primary splits. That is to say, we would obtain double the quantity of air with the same power when we double the number of splits, and likewise for any number of primary splits the quantity would be in proportion to the number of splits. But since all the circulation must be crowded through the main airway at a high velocity till the point is reached where the first split is made, we do not obtain an increase of quantity in the same proportion as we increase the number of splits. The increase in quantity will always be in a less ratio. To make this clear, let us take for illustration some practical examples.

EXAMPLE 1.—Find the power (foot-pounds per minute) that will circulate 24,000 cubic feet of air per minute in a mine, under the following conditions: the air to be circulated in one continuous current through an airway 16,000 feet long, including the return, the size of all the airways to be 6' × 10' throughout the mine.

SOLUTION.—Using formula **u'**,

$$u = \frac{k\,s\,q^3}{a^3} = \frac{.0000000217 \times 2\,(6 + 10) \times 16,000 \times 24,000^3}{(6 \times 10)^3} = 711,066 \text{ ft.-lb.}$$
per minute. Ans.

Example 2.—Find the power (foot-pounds per minute) that will circulate 24,000 cubic feet of air through a mine under the following conditions: Referring to Fig. 141, the air is divided at the foot of the downcast *d* into four splits, each 3,600 feet long and 6×10 feet in section, and finally united at the foot of the upcast *u*. The shafts are each 800 feet deep and are also $6' \times 10'$ in section. It will be noticed that the size and total length of airways are the same as in Example 1.

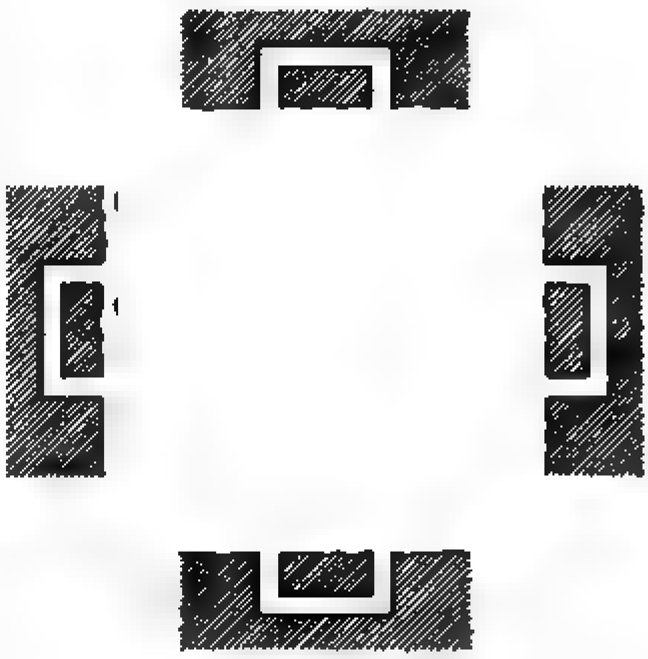


Fig. 141.

Solution.—It is evident that the total power is equal to the sum of the powers absorbed in the different passages. Using formula **u'**, the power absorbed in the two shafts is

$$u = \frac{k\,s\,q^3}{a^3} = \frac{.0000000217 \times 2\,(6 + 10) \times 2 \times 800 \times 24{,}000^3}{(6 \times 10)^3} = 71{,}107 \text{ ft.-lb.}$$

per minute, nearly.

Using the same formula and noting that, since the splits are equal, the quantity of air passing through each will be $\frac{24{,}000}{4} = 6{,}000$ cu. ft., the power absorbed by the splits is

$$u = \frac{k\,s\,q^3}{a^3} = \frac{.0000000217 \times 2\,(6 + 10) \times 4 \times 3{,}600 \times 6{,}000^3}{(6 \times 10)^3} = 10{,}000 \text{ foot-}$$

pounds per minute, nearly.

The total power is therefore $71{,}107 + 10{,}000 = 81{,}107$ ft.-lb. per min. Ans.

This result shows that in this case it required, after splitting, but 11.4 per cent. of the power originally required to pass the same quantity of air through the mine.

Example 3.—Determine the quantity of air that the power used in Example 1 will pass through a mine having the same conditions as those in Example 2.

Solution.—Applying law **15**,

$$81{,}107 : 711{,}066 = 24{,}000^3 : q_1^3;$$

or, $q_1 = \sqrt[3]{\frac{711{,}066}{81{,}107}} \times 24{,}000 = 49{,}488$ cubic feet per minute, nearly. Ans.

Comparing the result in Example 3 with the quantity circulated in Example 1, it will be seen that, by splitting as in

Example 2, over twice as much air is passed through the mine with the same power.

986₂. In all cases of equal splitting at a distant point in the mine, the increased quantity of air put in circulation by the original power may be found by the formula

$$q_1 = \frac{n\,q}{\sqrt[3]{1 + \frac{l}{L}(n^3 - 1)}}, \qquad (55_2.)$$

in which q = original quantity;
q_1 = increased quantity;
n = number of splits;
l = length of airway from beginning of intake to point of splitting;
L = total length of original airway.

EXAMPLE.—In a drift mine 3,500 feet long, 50,000 cubic feet of air per minute are circulated in a continuous current. What quantity will the same power circulate if three splits are made at a point on the intake 1,000 feet from the drift mouth?

SOLUTION.—Applying formula **55₂**,

$$q_1 = \frac{n\,q}{\sqrt[3]{1 + \frac{l}{L}(n^3 - 1)}} = \frac{3 \times 50{,}000}{\sqrt[3]{1 + \frac{1{,}000}{3{,}500}(3^3 - 1)}} = 73{,}706 \text{ cu. ft.} \quad \text{Ans.}$$

986₃. If the method of ventilating a mine be changed from a continuous current to a number of splits, the total quantity of air that the original power will pass through the splits can be found by the following formula, in which q_t = total quantity passing through the splits; q = quantity passing through original airway; a_t = total area of splits; a = area of original airway; s_t = total rubbing surface of splits; and s = rubbing surface of original airway. If desired, the quantities passing through the separate splits can then be found by the method used in Art. **992.**

$$q_t = \frac{a_t\,q}{a}\sqrt[3]{\frac{s}{s_t}}. \qquad (55_3.)$$

EXAMPLE.—If a certain power circulates 80,000 cu. ft. of air per min. through an airway 9′ × 6′ in section and 9,400 ft. long, what quantity will

it pass through the following splits which are substituted for the original airway? Split A, $9' \times 6'$ in section and 5,400 ft. long; split B $8' \times 5$ in section and 3,600 ft. long; split C, $6' \times 6'$ in section and 3,000 ft. long.

SOLUTION.—Using formula **55**$_8$,

$$q_t = \frac{(9 \times 6 + 8 \times 5 + 6 \times 6) \times 80{,}000}{9 \times 6} \times$$

$$\sqrt[3]{\frac{2(9+6) \times 9{,}400}{2(9+6) \times 5{,}400 + 2(8+5) \times 3{,}600 + 2(6+6) \times 3{,}000}} =$$

183,206, say, 183,200 cu. ft. per min. Ans.

987. To realize the benefits which may be obtained by splitting, consider Fig. 142, which is a simple, practicable case. D is the downcast and U the upcast shaft. Imagine the airways $A\,I$ and $K\,J$ to be removed. Then the air will flow down the shaft D and along the airway $D\,B\,C\,H\,E\,F\,G\,U$

FIG. 142.

and up the upcast shaft. Suppose that the distance $D\,B$ were 3,000 feet and the distance $A\,I$, 1,500 feet. The total distance traveled by the air from the foot of the downcast to the foot of the upcast would then be about 9,000 feet. Before the air reached F it would be foul and heavy with mine gases, carbonic acid gas, and other impurities, and it would be nearly impossible to work there. By splitting the air, this condition of things is remedied to a great extent. Thus, by splitting at A, fresh air from D passes along $A\,I$ and is again split at I; a part going to F and another part

towards E. At J, there is another split, a certain proportion going to K and the remainder to E, and from E to H. $D B$ is called the **main airway** and $U C$ the **main return.** After splitting at A, that portion of the air which does not pass along $L I$ continues along the main airway to the point C, where it passes into the main return and flows directly to the upcast at U. In order to accomplish the result described, a **bridge** is necessary at L to keep the fresh air from mingling with the return air; stoppings must be introduced at S, S, S, and a regulator must be placed at R. A **regulator** is an arrangement by which the sectional area of an airway can be reduced; it is virtually an increase of resistance to the movement of the flowing air. Only the reasons for using it will be mentioned here, as it will be described fully later.

Air or any other fluid will also travel along the path of least resistance, and always tends towards equilibrium. Now, suppose that it requires a greater power to force a certain quantity of air along the combined paths $A\ I\ F\ G\ U$, $I\ J\ K\ L$, and $J\ E\ H\ K$, than along the path $A\ B\ C\ U$; then more air will go towards B than towards I. But it is the exact opposite of this that is required; in other words, it is necessary that more air should flow towards I than towards B. By interposing a sufficiently great resistance at R, the greater volume of air will be forced to flow towards I.

This, then, is the principal object of splitting—*to supply the workings with fresh air.* Splitting must not, however, be carried to too great an extent, since every split reduces the velocity very rapidly, and the current will soon become too feeble to sweep out the noxious gases.

988. The first split, or, in fact, any split in the main airway, is variously called a **main split,** a **primary split,** or a **split of the first degree,** as at A. The second split, as at I, is called a **secondary split,** or a **split of the second degree.** The split at J is called a **tertiary split,** or a **split of the third degree.** It should be noted that

the degree of the split does not refer to the number of splits, though it happens so in the above case. The air coming along the main airway is divided at A, a part going to I; this part is again divided, a part going to J, and this last part is once more divided. Where two returns unite, as at G, K, and H, they are called **junctions.**

989. Unequal Splitting.—It was stated, when describing the necessity and action of a regulator, that the air always tended towards equilibrium. By this was meant that when the air had adjusted itself to the conditions governing its flow, a certain proportion would go one way and another proportion the other way, and no matter what the quantity passing might be, these proportions would always be preserved, provided there were no alterations in the lengths or sectional areas of the airways. To take a very simple illustration, suppose that in Fig. 143, D is the downcast and U the upcast.

FIG. 143.

Then, it is evident, from what has been previously stated, that more air will flow through $D\ C\ U$ than through $D\ B\ U$, since $D\ C\ U$ is shorter, has less rubbing surface, and, consequently, offers less resistance than $D\ B\ U$, the same sectional area and perimeter being assumed for both airways. In this case the air is split at D, and whenever there is a split in which one airway receives a greater quantity than the other, it is called an **unequal split.**

990. In every case of splitting, whether equal or unequal, *the pressure per square foot is the same in both splits.*

In order to explain this apparently inconsistent statement, one of the most important pertaining to the science of mine ventilation, it is necessary to digress for a time from the main subject.

In Fig. 144, let $A\ B\ D\ C$ represent a vessel filled with a

fluid, say water, for convenience, having two columns, $A\ B$ and $C\ D$, fitted with pistons, as shown, and communicating by the passageway $B\ D$. Suppose that the area of the smaller piston be 1 square foot and of the larger, 5 square feet; then, in order that there shall be equilibrium, that is, in order that the level of the water in both vessels shall be the same, the pressure per square foot on the piston at A must be equal to the pressure per square foot on the piston at C. This follows from **Pascal's law**, which states that *in the case of any fluid* (gas or liquid), *pressure is transmitted undiminished in all directions, whether downwards, upwards, or sideways.* If a force E of 5 pounds acts upon the piston A, a force of 5 pounds per square foot (since the area of A is 1 square foot) will be transmitted *upwards* against the piston C. Hence, to prevent C from moving upwards, a downward force F of $5 \times 5 = 25$ pounds must be applied to C. Moreover, it matters not what the areas of the pistons A and C are, the pressure per square foot must be the same on both pistons in order that they shall not move.

FIG. 144.

991. The same result obtains in a case like Fig. 145. Here a force E acts upon the piston A, and the pressure per square foot on A is transmitted with equal intensity to all parts of the surfaces touched by the water. This is exactly analogous to a split in which $A\ G$ represents the downcast shaft and $C\ G$ and $G\ H$ the splits.

As stated above, Pascal's law is true for either liquids or gases. It has to be modified somewhat when applying it to the case of air in motion, since it is then true only when the motion is uniform. But to secure uniformity

FIG. 145.

of motion, the resistance must be uniform, a condition which is practically always the case in mine ventilation. Since the airway up to the split requires a certain pressure to overcome the resistance it offers, this pressure should be deducted from the reading of the water-gauge, and the remainder treated as the pressure in each split.

992. Resuming now the subject of unequal splitting, consider Fig. 146. Let D and U represent the downcast and upcast shafts, respectively. Four unequal splits are here represented. The upcast and downcast shafts are $15' \times 10'$ and 600 feet deep; the airway $D\ A\ U$ is $5' \times 8'$ and 2,000 feet long; the airway $D\ B\ U$ is $6' \times 9'$ and 1,500 feet long; the airway $D\ C\ U$ is $7' \times 9'$ and 3,000 feet long, and the airway $D\ E\ U$ is $8' \times 10'$ and 1,800 feet long. Suppose that the velocity of the air in the shafts is 700 feet per minute and that it is required to find the pressure per square foot, the quantity passed by each split, and the horsepower required to circulate the air.

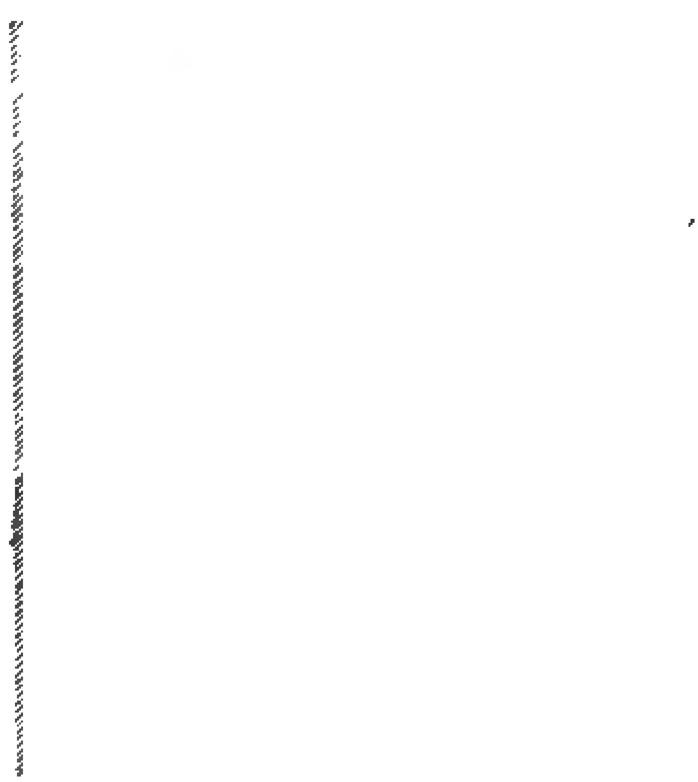


FIG. 146.

It is first necessary to find the total quantity of air passing through the shaft. This evidently equals, using formula **43**, $q = a\ v = 15 \times 10 \times 700 = 105{,}000$ cubic feet per minute.

The pressure per square foot required to pass this through the two shafts is

$$p = 2 \times \frac{k\ s\ v^2}{a} =$$

$$\frac{2 \times .0000000217 \times (2 \times 15 + 2 \times 10) \times 600 \times 700^2}{15 \times 10} =$$

4.2532 lb. per square foot.

Before the pressure per square foot for the splits can be found, it is necessary to calculate the quantities passing through each split. In order not to confuse the student, only the steps necessary for the calculation will here be given.

Let q_1, a_1 and s_1, q_2, a_2 and s_2, q_3, a_3 and s_3, q_4, a_4 and s_4 represent the quantity, sectional area, and rubbing surface in the splits *D A U*, *D B U*, *D C U*, and *D E U*, respectively. Then calculate the following expressions:

$$\sqrt{\frac{a_1^3}{s_1}} = \sqrt{\frac{40^3}{52,000}} = 1.1094, \text{ since } a_1 = 5 \times 8 = 40 \text{ and } s_1 = (2 \times 5 + 2 \times 8) \times 2,000 = 52,000.$$

$$\sqrt{\frac{a_2^3}{s_2}} = \sqrt{\frac{54^3}{45,000}} = 1.8706, \text{ since } a_2 = 6 \times 9 = 54 \text{ and } s_2 = (2 \times 6 + 2 \times 9) \times 1,500 = 45,000.$$

$$\sqrt{\frac{a_3^3}{s_3}} = \sqrt{\frac{63^3}{96,000}} = 1.6139, \text{ since } a_3 = 7 \times 9 = 63 \text{ and } s_3 = (2 \times 7 + 2 \times 9) \times 3,000 = 96,000$$

$$\sqrt{\frac{a_4^3}{s_4}} = \sqrt{\frac{80^3}{64,800}} = 2.8109, \text{ since } a_4 = 8 \times 10 = 80 \text{ and } s_4 = (2 \times 8 + 2 \times 10) \times 1,800 = 64,800.$$

$$sum = 7.4048$$

Dividing each of the above results by their sum, and multiplying by the total quantity passing through the shaft, 105,000 cubic feet per minute, the results thus obtained will be the quantities of air passing through the different splits. Thus,

$$q_1 = \frac{1.1094}{7.4048} \times 105,000 = 15,731 \text{ cu. ft. per minute in } DAU.$$

$$q_2 = \frac{1.8706}{7.4048} \times 105,000 = 26,525 \text{ cu. ft. per minute in } DBU.$$

$$q_3 = \frac{1.6139}{7.4048} \times 105,000 = 22,885 \text{ cu. ft. per minute in } DCU.$$

$$q_4 = \frac{2.8109}{7.4048} \times 105,000 = 39,859 \text{ cu. ft. per minute in } DEU.$$

$$sum = 105,000 \text{ cu. ft. per minute.}$$

Now, find the velocities by applying formula **44.**

$$v_1 = \frac{15,731}{40} = 393.3 \text{ ft. per minute in } DAU.$$

$$v_2 = \frac{26,525}{54} = 491.2 \text{ ft. per minute in } DBU.$$

$$v_3 = \frac{22,885}{63} = 363.3 \text{ ft. per minute in } DCU.$$

$$v_4 = \frac{39,859}{80} = 498.2 \text{ ft. per minute in } DEU.$$

Since the pressure is the same for each split, it is necessary to find it for one only. Hence,

$$p_1 = \frac{k s_1 v_1^2}{a_1} = \frac{.0000000217 \times 52,000 \times 393.3^2}{40} = 4.3637 \text{ lb. per}$$

square foot.

The total ventilating pressure per square foot is $4.2532 + 4.3637 = 8.6169$, say 8.62, pounds per square foot.

By formula **48**, the horsepower =

$$H = \frac{pq}{33,000} = \frac{8.62 \times 105,000}{33,000} = 27.43 \text{ horsepower, nearly.}$$

Examples similar to the above may be solved in the same way.

REGULATORS.

993. A regulator is shown in Fig. 147, and consists principally, as will be noticed, of a sliding shutter moving

Fig. 147.

in grooves. By means of this shutter, the width of the

opening may be adjusted so as to cause a greater or less quantity of air to pass through the airway.

In order to clearly understand the effect produced by a regulator, it is necessary to consider once more what determines the ventilating pressure per square foot. By formula **38**, $p = \frac{k s v^2}{a}$, and since the value of neither k nor a changes for the same airway through the introduction of a regulator, they may be neglected in *comparing* the results obtained by changing s and v. Now, if p represents the ventilating pressure per square foot in the spilts, it should be evident from what has been stated before that the mere introduction of a regulator in any split will not change the value of p in that split, provided the quantity of air passing through the other splits be not increased, since, if p were increased for one split, it would have to be increased a like amount also for the other splits, in order to restore the equilibrium according to Pascal's law, and this would increase the quantity of air in the other splits. But the introduction of a regulator in any split reduces the quantity of air passing through that split, and, as a consequence, reduces the velocity. Hence, if p is to remain the same, s must be increased, or some device must be used which will produce the same effect as increasing s; this device is the regulator itself. The conclusion is now evident: *the regulator is equivalent to lengthening the airway.*

994. Since, by formula **47**, the power $= u = p q$, and p remains the same after the regulator has been placed in the split, while q is reduced in consequence of the reduction of the quantity of air in the split containing the regulator, it is evident that less power will be required than before the regulator was introduced. Hence, if the power remains the same, both the velocity and the pressure will be increased throughout the mine, and the other splits will pass more air than before and *at a higher pressure.* This last is a very important feature in the case of gaseous mines, and will now be explained.

995. In Fig. 148, let D be the downcast and U the upcast shaft. The air is split at A, as shown. Suppose that the shafts are $8' \times 14'$ and 500 feet deep, and that all of the airways are $9' \times 12'$ and of the following lengths: $D\,A =$ 1,320 feet; $A\,B\,C\,U = 2{,}640$ feet $= \frac{1}{2}$ mile, and $A\,E\,U =$ 10,560 feet $= 2$ miles.

Suppose that the water-gauge in one of the return airways near U indicates, say, 1.53 inches, then the quantity of air passing in each split may readily be found. Since $p = 5.2\,W$, $p = 5.2 \times 1.53 = 7.956$ lb. per square foot. A certain amount of this is absorbed in overcoming the resistance of $D\,A$, while the remainder urges the air through

FIG. 148.

$A\,B\,C\,U$ and $A\,E\,U$. In order to find what proportion of the pressure is expended in $D\,A$, and what proportion in the splits, it is first necessary to find the relative velocities in the two splits. Representing by v_1 and v_2 the velocities of air in $A\,B\,C\,U$ and $A\,E\,U$, respectively, and applying law **(13)**,

$$v_1 : v_2 :: \sqrt{2} : \sqrt{.5}; \text{ or, } v_1 = v_2 \times \frac{\sqrt{2}}{\sqrt{.5}} = v_2 \times \sqrt{4} = 2\,v_2.$$

Now, representing by p_1 the pressure in the splits; by p, the pressure required to pass the air through the airway $D\,A$, and by v, the velocity in $D\,A$, we have, by applying formula **38**,

$$p = \frac{k\,s\,v^2}{a}, \text{ and } p_1 = \frac{k\,s_1\,v_1^2}{a};$$

in other words, $$p : p_1 :: \frac{k\,s\,v^2}{a} : \frac{k\,s_1\,v_1^2}{a};$$

or, $$p : p_1 :: s\,v^2 : s_1\,v_1^2.$$

996. Since both splits have the same sectional area and the velocity in the short one is twice that in the long one, it is evident that the short split passes twice the quantity that the long one does, or the short split passes two-thirds and the long split one-third of the total quantity coming along the airway DA. If v is the velocity in DA, it is evident (since the sectional areas are equal) that $v_1 = \frac{2}{3} v$, and $v_2 = \frac{1}{3} v$. Then, since $p : p_1 :: s v^2 : s_1 v_1^2$, $p : p_1 :: s v^2 : s_1 (\frac{2}{3} v)^2$; or, substituting the values of s and s_1, $p : p_1 :: (2 \times 9 + 2 \times 12) \times 1{,}320 \times v^2 : (2 \times 9 + 2 \times 12) \times 2{,}640 \times \frac{4}{9} v^2$; whence, $p = 1\frac{1}{8} p_1$.

Now, since $p + p_1 = 7.956$, $1\frac{1}{8} p_1 + p_1 = 7.956$, or $2\frac{1}{8} p_1 = 7.956$, and $p_1 = 3.744$ lb. per square foot = pressure for the splits. Also, $p = 1\frac{1}{8} p_1 = 1\frac{1}{8} \times 3.744 = 4.212$ lb. per square foot = pressure for DA.

Applying now formula **40**, the velocity in $A\,E\,U = v_2 =$

$$\sqrt{\frac{p_1 a}{k s}} = \sqrt{\frac{3.744 \times 9 \times 12}{.0000000217 \times (\overline{2 \times 9 + 2 \times 12} \times 2 \times 5{,}280)}} =$$

205 ft. per minute, very nearly.

Hence, $v_1 = 2 v_2 = 2 \times 205 = 410$ ft. per minute, and $v = 3 v_1 = 615$ ft. per minute. The quantity passing through $DA = q = a v = 9 \times 12 \times 615 = 66{,}420$ cu. ft. per minute. The quantity passing through the short split is $66{,}420 \times \frac{2}{3} = 44{,}280$ cu. ft. per minute, and through the long split, $66{,}420 \times \frac{1}{3} = 22{,}140$ cu. ft. per minute.

Applying formula **44** to find the velocity in the shaft,

$$v_s = \frac{q}{a_s} = \frac{66{,}420}{8 \times 14} = 593.04 \text{ ft. per minute,}$$

letting v_s, a_s, and p_s be the velocity, area, and pressure for the shaft, respectively. Remembering that there are two shafts, the pressure required to drive the air through them is

$$p_s = \frac{k s_s v_s^2}{a_s} =$$

$$\frac{.0000000217 \times (\overline{2 \times 8 + 2 \times 14} \times 500 \times 2) \times 593.04^2}{8 \times 14} =$$

2.998 lb. per square foot.

Consequently, the total pressure per square foot required to move the air is $7.956 + 2.998 = 10.954$ lb. per square foot; the power $= pq = 10.954 \times 66,420 = 727,565$ ft. lb. per minute, and the horsepower $= \frac{727,565}{33,000} = 22.05$ H. P.

997. It will be noticed that the velocity of the air in the long split *A E U* is very low, being but 205 feet per minute, and should the grade be an upward one, or even should there be no grade at all, it will be very difficult, if not impossible, to drive out any mine gas that may collect at *E*. To increase the power sufficiently to accomplish this would be a *very* costly method; but by putting a regulator at *R*, the quantity of air going through the short split may be so much reduced that with the same power a sufficient quantity of air may be driven through the long split as to dislodge the mine gases at *E*. If necessary, all of the air going through the short split may be shut off and the whole ventilative power of the mine applied to the long split. This is the most important result achieved by the regulator.

998. Suppose, however, that it was desired to ascertain the area of the regulator opening, in order to have the short split pass the same quantity of air that the long split passes. Taking the velocity in the long split as 205 feet per minute, that in the short split will then be 205 feet also, and the pressure required may be found by means of law (4) as follows: $p : p_1 :: v^2 : v_1^2$, or $3.744 : p_1 :: 410^2 : 205^2$; whence, $p_1 = .936$ lb. per square foot = pressure required to send the air through the split *A B C U* at a speed of 205 feet per minute. But the actual pressure is 3.744; hence, the regulator must offer a resistance of $3,744 - .936 = 2.808$ lb. per square foot. Assuming the regulator to have been adjusted properly, a water-gauge placed in it will show a difference of pressure between the two sides of the regulator of 2.808 lb. per square foot $= \frac{2.808}{5.2} = .54$ in. of water.

The area of the opening may now be calculated by aid of the following formula:

$$A = \frac{.0004\,q}{\sqrt{W}}, \qquad \textbf{(56.)}$$

in which A = area of opening in square feet;
q = quantity of air in cubic feet per minute which it is desired to pass through the opening;
W = difference of pressure in inches of water on the two sides of the regulator.

Substituting in formula **56** the values previously found,

$$A = \frac{.0004 \times 22{,}140}{\sqrt{.54}} = 12.05 \text{ sq. ft.}$$

The total quantity of air now going through the mine is $22{,}140 + 22{,}140 = 44{,}280$ cu. ft. per minute, or two-thirds of the quantity which went through before the regulator was introduced; and since the pressure per square foot remains the same as before, the horsepower required is but two-thirds of that previously required. Hence, if the horsepower be increased to its former value, the quantity will also be increased, but not to the same amount as before, since any increase in the quantity increases the velocity, which necessarily increases the frictional resistances—in other words, the ventilating pressure. The calculation will not be gone through with here to show just how much the ventilating pressure will be increased, as it is of no particular value to the student, and might tend to confuse him. He should, however, be able to see that the ventilating pressure and the velocity are both increased by the introduction of a regulator, and this is what is required to drive out the gas.

999. One more advantage obtained by splitting the air will now be noticed, and it is one of great value.

Fig. 149 represents a system of splits in which FA represents the fresh, or main, airway, and RA the return airway. The student will notice that when two arrow-heads are joined to one tail, there is a split, and when two tails

are joined to one head there is a junction. Suppose that in the left-hand half of the mine represented in the figure, gas were to accumulate in one of the farther workings, and the air had not sufficient pressure to drive it out. By shutting off the air in the other half of the mine, the entire power of

FIG. 149.

the ventilation may be employed to increase the pressure of the air in the left-hand half and drive out the gas. This is termed "sweeping out the mine," and is one of the greatest advantages obtained by splitting.

THE FIFTH ROOT.

1000. By aid of Table 25 the fifth root of any number may be found correctly to four figures. The arithmetical method of extracting the fifth root is very long and laborious. Since four figures are sufficient for all practical purposes in problems pertaining to mine ventilation, it was thought better to give the table than to give the rule generally used. The method of using the table will be exhibited by examples.

EXAMPLE.—Extract the fifth root of 1,264.782.

SOLUTION.—Only the first five figures of the number are required when using the table. When the sixth figure is 5, or greater, increase the fifth figure by 1, and omit the remaining figures. Doing so, the question becomes $\sqrt[5]{1,264.8} = ?$ Looking in column 4 of the table for

the nearest number *smaller* than the given number, it is found to be 1,158.6, opposite the number 4.1 in column 1, and 4.1 are the first two figures of the root. To find two more figures of the root, proceed as follows: $1,264.8 - 1,158.6 = 106.2$. Divide this remainder by the number in column 3 in the same row as the two numbers previously found, in this case 141.3, and obtain two figures of the quotient. If the second figure is greater than 5, increase the first figure by 1 and neglect the second figure. Should the second figure be a 5, obtain three figures of the quotient, and if the *third* figure is 5, or greater, increase the first figure by 1, and neglect the other two. Thus, $106.2 \div 141.3 = .751$, and the number to be used is .7, since the third figure is less than 5. It is necessary to obtain three figures of this quotient only when the *second figure is a 5.* Now, multiply this quotient, .7 in this case, by the number in column 2 and in the same row as the three previous numbers found in the table, and add the result to the number found in column 3. Thus, $6.89 \times .7 = 4.823$, and $141.3 + 4.823 = 146.123$. Finally, divide the difference found above (106.2) by 146.123; the result will be the next two figures of the root. Thus, $106.2 \div 146.123 = .727$, or .73. Hence, the entire root to four figures is 4.173. Ans.

EXAMPLE.—Find the fifth root of 45,261.

SOLUTION.—Only the numerical work is given; the student should read the explanation given above in connection with the work. $45,261 - 44,371 = 890$. $890 \div 2,610 = .34$, or .3. $61.4 \times .3 = 18.42$. $2,610 + 18.42 = 2,628.42$. $890 \div 2,628.42 = .338$, say .34. Hence, $\sqrt[5]{45,261} = 8.534$. Ans.

If the number is wholly decimal, take the first five figures to the right of the decimal point (annexing ciphers if necessary to make five figures) and treat the number as if it were a whole number with five figures.

EXAMPLE.— $\sqrt[5]{.664} = ?$

SOLUTION.—Annexing two ciphers to make the necessary five figures, $\sqrt[5]{.664} = \sqrt[5]{.66400}$. Whence, $66,400 - 65,908 = 492$. $492 \div 3,582 = .13$. $77.9 \times .1 = 7.79$. $3,582 + 7.79 = 3,589.79$. $492 \div 3,589.79 = .137$, or .14. Hence, $\sqrt[5]{.664} = .9214$. Ans.

EXAMPLE.— $\sqrt[5]{42,675,830} = ?$

SOLUTION.—Begin at units place and point off the number into periods of **five** figures each. Thus, 426'75830. Retain the first five figures, beginning with the left, the result is 426'76. Regarding the division mark for the present as a decimal point, proceed as in the preceding examples. $426.76 - 391.35 = 35.41$. $35.41 \div 59.3 = .59$, or .6. $3.59 \times .6 = 2.154$. $59.3 + 2.154 = 61.454$. $35.41 \div 61.454 = .576$, or .58. Hence, the figures of the root are 3358. The position of the decimal

point may be determined from the statement that *there must be as many figures in the integral part of the root as there are periods in the integral part of the number whose root is to be found.* Since there are two such periods in the above number, $\sqrt[5]{42,675,830} = 33.58$.

Ans.

Had the number been 4,267,583,000,000, the number of periods would have been three, and the fifth root,

$$\sqrt[5]{426'75830'00000} = 335.8.$$

It will be a good exercise for the student to prove the following:

$$\sqrt[5]{426,758.3} = 13.37; \sqrt[5]{4,267,583} = 21.19;$$

$$\sqrt[5]{426,758,300} = 53.22, \text{ and } \sqrt[5]{4,267,583,000} = 84.84.$$

1001. If it is absolutely necessary for the student to extract the fifth root without the aid of a table, he may do so in the following manner:

$\sqrt[5]{4,267,583} = ?$

1. Point off the number into periods, as above directed, obtaining in this case 42'67583.

2. Find a number expressed by one figure whose fifth power is next less than the number expressed by the first period. It will aid the student, in finding the first figure of the root, if he will remember that if the first period contains but one figure, the first figure of the root must be 1; if but two figures, the first figure of the root can not be greater than 2; if but three figures, the first figure of the root is either 2 or 3; if but four figures, the first figure of the root can not be greater than 6, and if the first period contains five figures, the first figure of the root may be 6, 7, 8, or 9. Try 2 for the first figure of the root of the above number and raise it to the fifth power; the result is $2^5 = 32$. Since 32 is less than 42, the first figure of the root is 2.

3. To find the second figure, subtract the fifth power of the first figure from the first period and annex the second period to the remainder, or, if there is no second period, bring down five ciphers. Performing the operation on the above number, $42 - 32 = 10$; annexing the second period, the result is 1,067,583.

4. Raise the first figure of the root to the fourth power,

multiply the result by 5, and annex four ciphers. Annex four ciphers to the cube of the first figure, and add the result to the last result. Thus, $2^4 \times 5 = 80$; annexing four ciphers $= 800,000$. 2^3 with four ciphers annexed $= 80,000$, and $800,000 + 80,000 = 880,000$.

5. Divide the result obtained in 3 by the result obtained in 4, and the quotient will *very probably* be the second figure of the root. Thus, $1,067,583 \div 880,000 = 1 +$, and the first two figures of the root are 21.

6. Raise the first two figures of the root to the fifth power and subtract the result from the given number whose root is to be found, annexing five ciphers to the given number if it contains but one period. Thus, $21 \times 21 = 441$; $441 \times 21 = 9,261$, the cube; $9,261 \times 21 = 194,481$, the fourth power, and $194,481 \times 21 = 4,084,101$. Hence, $4,267,583 - 4,084,101 = 183,482$.

7. Multiply the fourth power of the first two figures (obtained in 6) by 5, and divide the remainder obtained in 6 by the result, and obtain two figures of the quotient. If the second figure of the quotient is greater than 5, increase the first figure by 1 and neglect the second figure; otherwise, use only the first figure. Should the second figure be 5, obtain three figures of the quotient, and if the *third* figure is 5, or greater, increase the first figure by 1. Thus, $21^4 = 194,481$ (see 6), and $194,481 \times 5 = 972,405$; then, $183,482 \div 972,405 = .18 +$ or .2.

8. Multiply the cube of the first two figures of the root (obtained in 6), with a cipher annexed, by the number found in 7, and add the result to 5 times the fourth power (obtained in 7). Thus, $21^3 = 9,261 = 92,610$, with a cipher annexed. $92,610 \times .2 = 18,522$. $972,405 + 18,522 = 990,927$.

9. Divide the remainder obtained in 6 by the result obtained in 8 and carry the quotient to three *decimal* places. If the third figure of the decimal is 5 or greater, increase the second figure by 1. These two figures of the quotient are the third and fourth figures of the root. Thus, $183,482 \div 990,927 = .185$, say .19. Hence the figures of the root

are 2119, and since there are two periods, $\sqrt[5]{4,267,583} = 21.19$. Ans.

NOTE.—The method outlined above is exactly what is accomplished by means of Table 25, but the work is very much more laborious. It is, however, the simplest known method of finding the fifth root of numbers.

EXAMPLE.— $\sqrt[5]{9} = ?$

SOLUTION—2. Since there is but one figure, the first figure of the root is 1.

3. $9 - 1^5 = 8$, since $1^5 = 1$. Annexing five ciphers gives 800,000.

4. $1^4 \times 5$ with four ciphers annexed $= 50,000$; 1^3 with four ciphers annexed $= 10,000$; the sum $= 50,000 + 10,000 = 60,000$.

5. $800,000 \div 60,000 = 13 +$. This result is much too high, since the quotient thus obtained (which is the probable second figure of the root) should not exceed 9. Now, remembering that the fifth power of 2 is 32, it is evident that $\sqrt[5]{9}$ must be considerably less than 1.9, which nearly equals 2. Trying 1.6, the fifth power is $1.6^5 = 10.48576$, which is also too high, but quite close; hence, 1.5 is probably the correct number to use, and the first two figures of the root are 15.

6. $15 \times 15 = 225$; $225 \times 15 = 3,375$; $3,375 \times 15 = 50,625$, and $50,625 \times 15 = 759,375$. $900,000 - 759,375 = 140,625$.

7. $15^4 \times 5 = 50,625 \times 5 = 253,125$; $140,625 \div 253,125 = .555$, or .6.

8. 15^3 with a cipher annexed $= 33,750$; $33,750 \times .6 = 20,250$, and $253,125 + 20,250 = 273,375$.

9. $140,625 \div 273,375 = .514$, say .51. Hence, $\sqrt[5]{9} = 1.551$. Ans.

TABLE 25.

1	2	3	4	1	2	3	4
1.0	.100	.5000	1.0000	5.6	17.6	491.7	5,507.3
1.1	.133	.7321	1.6105	5.7	18.5	527.8	6,016.9
1.2	.173	1.037	2.4883	5.8	19.5	565.8	6,563.6
1.3	.220	1.428	3.7129	5.9	20.5	605.9	7,149.2
1.4	.274	1.921	5.3782	6.0	21.6	648.0	7,776.0
1.5	.338	2.531	7.5938	6.1	22.7	692.3	8,446.0
1.6	.410	3.277	10.486	6.2	23.8	738.8	9,161.3
1.7	.491	4.176	14.199	6.3	25.0	787.6	9,924.4
1.8	.583	5.249	18.896	6.4	26.2	838.9	10,737
1.9	.686	6.516	24.761	6.5	27.5	892.5	11,603
2.0	.800	8.000	32.000	6.6	28.7	948.7	12,523
2.1	.926	9.724	40.841	6.7	30.1	1,007	13,501
2.2	1.06	11.71	51.536	6.8	31.4	1,069	14,539
2.3	1.22	13.99	64.363	6.9	32.9	1,133	15,640
2.4	1.38	16.59	79.626	7.0	34.3	1,201	16,807
2.5	1.56	19.53	97.656	7.1	35.8	1,271	18,042
2.6	1.76	22.85	118.81	7.2	37.3	1,344	19,349
2.7	1.97	26.57	143.49	7.3	38.9	1,420	20,731
2.8	2.20	30.73	172.10	7.4	40.5	1,499	22,190
2.9	2.44	35.36	205.11	7.5	42.2	1,582	23,730
3.0	2.70	40.50	243.00	7.6	43.9	1,668	25,355
3.1	2.98	46.18	286.29	7.7	45.7	1,758	27,068
3.2	3.28	52.43	335.54	7.8	47.5	1,851	28,872
3.3	3.59	59.30	391.35	7.9	49.3	1,948	30,771
3.4	3.93	66.82	454.35	8.0	51.2	2,048	32,768
3.5	4.29	75.03	525.22	8.1	53.1	2,152	34,868
3.6	4.67	83.98	604.66	8.2	55.1	2,261	37,074
3.7	5.07	93.71	693.44	8.3	57.2	2,373	39,390
3.8	5.49	104.3	792.35	8.4	59.3	2,489	41,821
3.9	5.93	115.7	902.24	8.5	61.4	2,610	44,371
4.0	6.40	128.0	1,024.0	8.6	63.6	2,735	47,043
4.1	6.89	141.3	1,158.6	8.7	65.9	2,864	49,842
4.2	7.41	155.6	1,306.9	8.8	68.1	2,998	52,773
4.3	7.95	170.9	1,470.1	8.9	70.5	3,137	55,841
4.4	8.52	187.4	1,649.2	9.0	72.9	3,281	59,049
4.5	9.11	205.0	1,845.3	9.1	75.4	3,429	62,403
4.6	9.73	223.9	2,059.6	9.2	77.9	3,582	65,908
4.7	10.4	244.0	2,293.5	9.3	80.4	3,740	69,569
4.8	11.1	265.4	2,548.0	9.4	83.1	3,904	73,390
4.9	11.8	288.2	2,824.8	9.5	85.7	4,073	77,378
5.0	12.5	312.5	3,125.0	9.6	88.5	4,247	81,537
5.1	13.3	338.3	3,450.3	9.7	91.3	4,426	85,873
5.2	14.1	365.6	3,802.1	9.8	94.1	4,612	90,392
5.3	14.9	394.5	4,182.0	9.9	97.0	4,803	95,099
5.4	15.7	425.2	4,591.7	10.0	100.0	5,000	100,000
5.5	16.6	457.5	5,032.8				

MINE VENTILATION.

(PART 2.)

THE PRODUCTION OF VENTILATING CURRENTS.

VARIOUS SYSTEMS OF INDUCING CURRENTS.

1002. Ventilating Currents.—The motion of air-currents in mines is caused by a difference in pressure between the two ends of the current, or, in other words, a difference in pressure between the downcast and upcast. The direction of the flow is always from the higher towards the lower pressure.

In the case of ventilation produced by exhaust-fans or furnaces, the higher pressure is the normal pressure of the atmosphere, and the lower pressure is that produced in the fan-drift, or at the bottom of the furnace-shaft. In the case of a blowing-fan, the higher pressure consists of the atmospheric pressure plus the pressure exerted by the fan, and the lower pressure is the atmospheric pressure at the top of the upcast. A waterfall in the downcast shaft produces motion in a current on the same principle as a blowing-fan, and a steam-jet in the upcast acts on the same principle as the exhaust-fan. However, it must be borne in mind that neither of the two latter methods is as efficient as a fan. These facts show clearly that the object of all artificial ventilating appliances must be to provide the required difference of pressure. Current motion may, therefore, be caused by either of two methods: (*a*) methods of compression, by means of which the air in the downcast is raised to a pressure greater than the atmosphere, or (*b*) methods of

exhaustion, by means of which the pressure of the air in the upcast is made less than the pressure of the atmosphere

1003. The Laws of Current Motion.—As a current of air for mine ventilation begins and ends in the atmosphere, it is necessary that a ventilator be applied to produce a terminal depression for the current to fall into and a subsequent compression to finally force it out into the

FIG. 150.

atmosphere. In Fig. 150 a pump is used to illustrate what has been expressed in words. The water in the cistern A is subject to the pressure of the atmosphere, and falls into a depression at B, through the pipe M. The depression at B is created by the pump in the same manner as a fan creates a depression. The excess of pressure in the atmosphere over that exerted at B is the measure of depression. It is this depression that causes the water to flow from A to B, and a similar depression that causes the air in a mine to flow

from the top of the downcast into the upcast. In the illustration, the pump *P* produces the depression. The lifting of the piston reduces the pressure of the atmosphere on the water in *B*, and the falling water pressed by the atmosphere at *A* rushes in to fill up the void. Without this depression, the water would naturally rise in the pump to the level of *A*, but would remain at rest and not flow out. Therefore, further energy is required to cause it to rise high enough to flow out of the nozzle. In the same way a fan must not only cause a depression, so as to cause the air to flow into the fan-drift, but it must also exert energy to force the air out into the atmosphere. In the case of the pump, to raise the water to *L*, and enable it to flow out of the nozzle, energy equal to a fall from *L* to the nozzle is required. This fall overcomes the friction and the delivery pressure, and is similar to the compression required in an exhaust-fan for it to throw the mine current out of its chimney. The illustration shows clearly that *D* is the measure of the depression below the pressure of the atmosphere, and that *C* is the measure of the compression above the atmosphere; further, it explains the principles of the double fall, or the fall from the atmospheric pressure at *A* to the depression at *B*, and the fall from the pressure above the atmosphere at *L* to the atmospheric pressure at *A*.

1004. How Ventilating Currents Are Produced.—The means by which ventilating currents are produced are all included under the following heads:

(*a*) Ventilation by natural heat.
(*b*) Ventilation by artificial heat.
(*c*) Ventilation by waterfalling.
(*d*) Ventilation by mechanical agencies.
(*e*) Ventilation by a steam-jet.
(*f*) Ventilation by a water-jet.

1005. Natural ventilation is produced in a mine when the top of the upcast and the top of the downcast are at different elevations, or, in other words, when one is some distance up a hill and the other at or near the base. A

natural ventilating current is only set in motion when the temperature of the outer air and that of the walls of the mine passages is different. This method of ventilation differs from all others in one important respect, namely, the direction of the current is reversed in summer from what it is in winter. In summer, when the external air is hotter than the walls of the mine passages, the warm air descends the deeper shaft, and in so doing is cooled by the absorption of heat by the walls of the shaft. This cooled column thus becomes a heavier one than the one parallel to it, shown in (*a*), Fig. 151. In this figure, *a b* is the shaft and *m* is the mine. The cooled air column in *a b*, being heavier than the external column *d c*, causes the air to flow from *b* to *d*. The direction of flow in winter is illustrated in (*b*), Fig. 151.

(*a*)

FIG. 151.

As the walls of the mine passages are warmer than the external air, the column of air in the shaft *a b* is warmer than the parallel column of the external air *c d*. Therefore, the external column being heavier forces the warmer, lighter column up the shaft, and causes the current to flow from *d* to *b*.

In the event of the external air having the same temperature as the walls of the mine passages, there is no flow of air-current, because one column balances the other.

1006. Ventilation by artificial heat is produced by a furnace fire situated at the bottom of the upcast shaft. This fire heats the column of air in the upcast shaft and makes it less dense and lighter than the column of cold air

in the downcast. The weight of a cubic foot of air in either shaft is calculated by formula **57**,

$$W = \frac{1.3253 \times B}{(459 + t)}; \qquad \textbf{(57.)}$$

B = the barometic pressure in inches of mercury;
t = Fahrenheit temperature of air in the shaft;
W = weight of a cubic foot of air.

Example.—The downcast shaft of a mine is 600 feet deep the mean barometric pressure in the shaft is 30 inches, and the mean temperature of the air in the shaft is 62° F. What is the average weight of a cubic foot of air in this shaft?

Solution.—Applying formula **57**,

$$W = \frac{1.3253 \times 30}{(459 + 62)} = .07631 \text{ lb. Ans.}$$

Example.—The upcast shaft of the same mine is 600 feet deep; the mean barometric pressure is the same (30 inches), and the mean temperature of the air in the shaft is 196° F. What is the average weight of a cubic foot of air in this shaft?

Solution.—Applying formula **57**,

$$W = \frac{1.3253 \times 30}{(459 + 196)} = .0607 \text{ lb. Ans.}$$

The two foregoing examples show that the air in the upcast shaft is lighter than that in the downcast.

Now, if the height of each column is taken at 600 feet, the ventilating pressure per square foot can be found by multiplying the weight of air per cubic foot by the height of the column in feet. Thus,

$$\begin{aligned} \text{Downcast} &= .07631 \times 600 = 45.786 \text{ lb.} \\ \text{Upcast} &= .0607 \times 600 = 36.420 \text{ lb.} \end{aligned}$$

Difference, or ventilating pressure per sq. ft. = 9.366 lb.

EXAMPLES FOR PRACTICE.

1. The downcast shaft of a mine is 437 feet deep, the mean barometric pressure is 30.25 inches, and the mean temperature of the air in the shaft is 67° F. What is the weight of a column of air in this shaft, having a base of 1 square foot? Ans. 33.3081 lb

2. The downcast shaft of a mine is 1,147 feet deep, the mean barometric pressure is 29.9 inches, and the mean temperature of the air in

the shaft is 50° F. What is the weight of a column of air in this shaft, having a base of 1 square foot? Ans. 89.29895 lb.

3. The upcast shaft of a mine is 347 feet deep, the mean barometric pressure is 30 inches, and the mean temperature of the air in the shaft is 187° F. What is the weight of a column of air in this shaft, having a base of 1 square foot? Ans. 21.35785 lb.

4. The upcast shaft of a mine is 1,170 feet deep, the mean barometric pressure is 29.5 inches, and the mean temperature of the air in the shaft is 160° F. What is the weight of a column of air in this shaft, having a base of 1 square foot? Ans. 73.8972 lb.

EFFECT OF TEMPERATURE ON VOLUME.

1007. The volume of a given quantity of air varies directly as its absolute temperature, the barometric pressure and weight remaining the same. This principle is expressed in formula **58**,

$$T = \frac{V}{v} \times (459 + t), \qquad \textbf{(58.)}$$

in which T = absolute temperature of greater volume;
V = greater volume;
v = lesser volume;
and t = given temperature of lesser volume in Fahrenheit degrees.

EXAMPLE.—If the volume of a given quantity of air is 35,672 cubic feet when its temperature is 57° F., what must its temperature be to increase the volume to 51,756 cubic feet, supposing the atmospheric pressure and weight to remain the same?

SOLUTION.—Applying formula **58**, $\frac{51,756}{35,672} \times (459 + 57) = 748.65°$, or the absolute temperature necessary for increasing the volume. Now, having the absolute temperature, it is necessary to reduce it to Fahrenheit temperature. This can be readily done by subtracting 459 from the absolute temperature. Then, 748.65° − 459° = 289.65° F. Ans.

EXAMPLES FOR PRACTICE.

1. If the volume of a given quantity of air when its temperature is 60° F. is 46,732 cubic feet, what must its temperature be to increase the volume to 65,000 cubic feet, supposing the atmospheric pressure and weight to remain the same? Ans. 262.88° F.

2. If the volume of a given quantity of air when its temperature is 160° F. is 65,000 cubic feet, what must its temperature be when the volume is 50,000 cubic feet, supposing the atmospheric pressure and weight to remain the same? Ans. 17.1° F.

THE MOTIVE COLUMN.

1008. The ventilating pressure can be found directly through the medium of the motive column. This motive column is the short column of air whose weight provides the ventilating pressure. If the length of this motive column is subtracted from the length of the downcast column, the weight of the remaining portion of the downcast column is equal to the weight of the upcast column. This is explained by Fig. 152, in which U is the upcast column and D is the downcast column; the furnace is shown at F, and $M\,C$ is the motive volume. This motive column, then, is a column of air in the downcast shaft whose weight is equal to the excess of weight of the cold column over that of the hot one. The most convenient way to find the length of the motive column is by what is called Nicholas Wood's formula, which is based on the law that the weights of the columns are inversely as their absolute temperatures. This law can be expressed by formula **59**,

FIG. 152.

$$M = \frac{(t - t_1)}{(459 + t)} \times D, \qquad (59.)$$

in which t = higher temperature, or that of the upcast;
t_1 = lower temperature, or that of the downcast;
D = depth of the shaft in feet;
and M = motive column.

Now suppose a case in which the mean temperature of

the downcast shaft is 58° F., and the mean temperature of the furnace shaft is 186° F., and the depth of the shafts is 800 feet. By formula **59**, the length of the motive column in this case will be equal to $\frac{(186 - 58)}{(459 + 186)} \times 800 = 158.75$ feet.

If, then, the average weight of a cubic foot of air in the downcast shaft is calculated by formula **57**, and is found to be .077 pound, the ventilating pressure can be found by the following formula:

$$p = \frac{(t - t_1)}{(459 + t)} \times .077 \times D, \qquad \textbf{(60.)}$$

in which p = ventilating pressure in pounds per square foot;
t = higher temperature;
t_1 = lower temperature;
and D = depth of shaft.

Thus, $\frac{(186 - 58)}{(459 + 186)} \times .077 \times 800 = 12.224$ pounds per square foot, the ventilating pressure required.

EXAMPLE.—The ventilating shafts of a mine are each 800 feet deep, the temperature of the downcast column is 58° F., and that of the upcast column is 202° F. What is the weight of a column of air in the downcast shaft 1 square foot in the base, and what is the weight of a column of equal length in the upcast shaft? Show by formula **60** that the difference between the weights of the two columns is equal to the weight of the motive column, the mean barometric pressure in the two shafts being 30.5 inches.

SOLUTION.—The weight of a cubic foot of air in the downcast shaft is, by formula **57**, equal to $\frac{1.3253 \times 30.5}{459 + 58} = .078185$ pound, and, by formula **57**, the weight of a cubic foot in the upcast shaft is found to be

$$\frac{1.3253 \times 30.5}{459 + 202} = .061152 \text{ lb. Ans.}$$

Having found the weight of a cubic foot of air in each shaft, the weight of a column with a base of 1 square foot in each shaft is found by multiplying the weight per cubic foot of air in each shaft by the depth of the shaft; therefore, the weight of the downcast column equals $.078185 \times 800 = 62.548$ pounds, and the weight of the upcast column equals $.061152 \times 800 = 48.922$ pounds. The difference between the respective weights of the columns $= 62.548 - 48.922 = 13.626$ pounds.

By formula **60**, if the weight of the cubic foot of air is taken at .078, as the barometer is high, $p = \frac{(202 - 58)}{(459 + 202)} \times .078 \times 800 = 13.594$ pounds.

It will be observed in this connection that the result secured by using formula **60** is a little less than that found by using the weights of the columns, but the difference arises entirely from the fact that the weight of a cubic foot in the downcast column is actually .078185 pound instead of .078; had the weight of a cubic foot of air been taken at .078185 pound, the answers would have agreed more closely.

EXAMPLES FOR PRACTICE.

NOTE.—The weight of 1 cubic foot of air at a temperature of 62° F., and a barometric pressure of 30 inches, is equal to .076 pound. This is close enough for the weight of a cubic foot of air for use in the following examples.

1. The ventilating shafts of a mine are each 950 feet deep, the temperature of the downcast column is 60° F., and that of the upcast is 230° F. (*a*) What is the length of the motive column? (*b*) What is the difference in the weights of the ventilating columns per square foot of area?

   Ans. { (*a*) 234.4 ft.
   (*b*) 17.814 lb.

2. The ventilating shafts of a mine are each 760 feet deep, the temperature of the downcast is 52° F., and that of the upcast is 280° F. (*a*) What is the length of the motive column? (*b*) What is the difference in the weights of the ventilating columns per square foot of area?

   Ans. { (*a*) 234.5 ft.
   (*b*) 17.82 lb.

VENTILATING BY FURNACES.

THE CONSTRUCTION OF FURNACES.

1009. As the furnace is still used in some regions for the ventilation of small mines where the output does not justify the erection of a ventilating fan, a few facts concerning its construction and use should be known. The object of a furnace is to produce a motive column by rarefying the air in the upcast shaft with heat. In shallow mines, however, where an efficient motive column can not be obtained,

the fan is much more efficient and economical. In spite of this, the furnace is still used. Therefore, its construction must be described.

Fig. 153 is an illustration of a type of furnace quite generally used. It is important, in building a furnace, to construct it so as to keep the excessive heat of the fire from the coal on its flanks, and from the rock above it. In Fig. 153, *L*, *L* show the sides in the coal-seam. The drifts *D*, *D* provide for the isolation of the heat from the coal. Immediately above the fire is a double arch, and as the inner one is subject to constant variations of temperature, ribs of brick are run between the inner and the outer arch to prevent collapse, and to keep the air-space so widely open that a current of air may freely pass through it and keep the heat from the roof. The importance of this arrangement is due to the fact that in cases where the roof stone contains water, the crown arch is continually buckling with the pressure produced by steam, and this causes the top stone to break and fall. *P* is the ash-pit, and *G* is the bearing-up bar, or front fire-grate girder. At *B* are seen the fire-bars that conjointly make up the fire-grate surface. The furnace arch is generally semicircular, and the height from the fire-bars to the under surface of the arch is generally $1\frac{1}{2}$ times the width of the fire-grate surface. The dimensions of the furnace are determined on the basis of the amount of work it is intended to perform, and when the breadth of the furnace is found, all the other dimensions are deduced from it.

FIG. 153.

The length of the furnace-bars should not exceed 5 feet, and as this dimension is uniform for all furnaces, the important dimension required for constructing a furnace is its breadth. The area of the fire-grate surface varies inversely as the square root of the depth of the furnace-shaft. Before the width of a furnace can be determined, the amount of air necessary for the efficient ventilation of the mine must be fixed, and, in addition to this, the ventilating pressure in inches of water-gauge must be approximately known. From these factors, the horsepower of the required furnace can be calculated by dividing the product of the volume of air in cubic feet per minute and the pressure in pounds per square foot, by 33,000.

EXAMPLE.—Suppose a case in which the quantity of air required is 120,000 cubic feet per minute, and the probable mine resistance for that quantity is 2 inches of water-gauge; what horsepower is required in the ventilation?

SOLUTION.— $\frac{120{,}000 \times 2 \times 5.2}{33{,}000} = 37.8$ H. P. Ans.

EXAMPLES FOR PRACTICE.

1. A mine is ventilated by an air-current of 200,000 cubic feet per minute, and the water-gauge reading is 2.1 inches; what horsepower is exerted in moving the air? Ans. 66.18 H. P.

2. A mine is ventilated by an air-current of 125,000 cubic feet per minute, and the water-gauge reading is 3.5 inches; what horsepower is exerted in moving the air? Ans. 68.94 H. P.

GRATE SURFACE.

1010. The fire-grate surface required is found by the following formula:

$$s = \frac{34}{\sqrt{D}}, \qquad (61.)$$

in which D = depth of the furnace-shaft in feet;
34 = a constant number proved by many experiments;
and s = square feet of fire-grate surface required per horsepower of the ventilation.

EXAMPLE.—The depth of the shaft is 400 feet, and the horsepower required in the ventilation is 37.8; what area of fire-grate is required?

SOLUTION.—By applying formula **61**, $\frac{34}{\sqrt{400}} = 1.7$ square feet of fire-grate surface required per horsepower. Since 37.8 horsepower is required, the fire-grate surface should be $37.8 \times 1.7 = 64.26$ sq. ft. Ans.

A grate surface of the size calculated in the above example will efficiently ventilate a mine 400 feet deep, with 120,000 cubic feet of air per minute, circulated against a resistance equal to 2 inches of water-gauge. Again, if the bars of the fire-grate are 5 feet long, the breadth of the furnace, in feet, will in this case be equal to $\frac{64.26}{5} = 12.85$ feet.

EXAMPLE.—Let a furnace-shaft be 900 feet deep, and the ventilating current be equal to 200,000 cubic feet per minute, with a mine resistance equal to 2 inches of water-gauge; what must be the breadth of the furnace when the length of the fire-bars is taken at 5 feet?

SOLUTION.—The horsepower required is equal to $\frac{200{,}000 \times 2 \times 5.2}{33{,}000} =$ 63 H. P. The fire-grate surface per horsepower, by use of formula **61**, is found to equal $\frac{34}{\sqrt{900}} = 1.133$ square feet; and, therefore, the square feet of fire-grate surface required are equal to $63 \times 1.133 = 71.379$ square feet; and, if the length of the fire-bars be taken at 5 feet, the breadth of the furnace is equal to $\frac{71.379}{5} = 14.28$ ft. Ans.

An examination of the two examples will show that, notwithstanding the fact that the horsepower required in the latter case is so much greater than in the former, yet the fire-grate surface is very little increased, owing to the greater depth of the shaft.

EXAMPLES FOR PRACTICE.

1. What grate surface will be required to produce a current of 200,000 cubic feet per minute, with a water-gauge of 2.1 inches, if the furnace-shaft is 900 feet deep? Ans. 74.98 sq. ft.

2. What width of furnace will be required to produce a current of 100,000 cubic feet per minute, with a water-gauge of 2 inches, if the shaft is 625 feet deep, and the grate-bars of the furnace are 5 feet long? Ans. 8.57 ft.

REMARKS ON FURNACE VENTILATION.

1011. Where furnace ventilation is practised and the return air contains inflammable gas, it is often necessary to feed the furnace with fresh air and use the heated gases from the fire to heat and rarefy the upgoing column of return air from the mine. In Fig. 154 the heated air from the furnace is marked *H*, and is seen to pass up that portion of the shaft at *S*. Again, nothing but the return air *R* is seen to pass the dumb drift *P*, as shown by the arrows. The return air unites and mixes with the heated gases of the fire at the junction of the dumb drift with the shaft. The furnace receives its supply of fresh air at *A*, as shown in section and plan. In the plan, the fresh air to feed the furnace is indicated by the arrow at *A*, and in the section the return air from the mine is seen to enter the dumb drift at *R*. The object of this drift is to isolate the return air from the flaming gases of the furnace. The junction of the dumb drift with the shaft should not occur at a less elevation than 150 feet above the furnace, and in some cases where bituminous coal is burned, safety is not secured until the junction takes place at an elevation of 300 feet. As this is equal to the depth of many shafts, and more than the maximum depths of others where furnaces are used, it is clear that, at its best, the furnace does not afford a safe means for the ventilation of a gaseous mine.

FIG. 154.

VENTILATION BY WATERFALLING.

1012. In some regions, waterfall ventilation is important, because it is cheap and very efficient. It often occurs, however, through oversight, that this agency is not adopted, although the conditions for its use are highly favorable. In Fig. 155 is shown what is called a **trompe,** or **waterfall ventilator,** in common use in many parts of the world, for the ventilation of such coal and metal mines as have the shaft bottom or lower level situated above the drainage level of the district. The trompe is a rectangular tube made of wood, and has an area of section equal to from 4 to 6 square feet. The length is regulated by the prevailing conditions, but the greater the length the better. The water delivered into the trompe generally comes from a neighboring stream and is conducted by a spout or trough *L*. Here the water is first divided into small streams by passing it through perforations in the top plate *G*. These water threads are broken up by their inert force into drops that fall in succession from one to another of a series of sloping shelves that are called dashboards, as shown by *D*, *D*, *D*, *D*, etc. The water, on striking the upper board, rebounds, and the spray that is thus produced rains on to the under ones, and so on from one to another until the trompe becomes a vertical tube enclosing a shower of fine water-drops that produce a powerful, energetic blast of air. As the drops fall they produce a partial vacuum in their rear and a compression on their under

FIG. 155.

side, which causes an inrush of air into the ports *A*, *A*, *A*, etc. The water ultimately falls into a trap *W T*, where it overflows, and the air then blows out of the horizontal delivery branch of the ventilator at *V*. The trompe is used in the downcast shaft, and, therefore, acts as a blower to propel a current through the galleries of the mine.

1013. Where copious ventilation is required, an entire shaft is made to act as a large trompe, as in the case illustrated by Fig. 156. Here, however, instead of using a perforated plate, a brush mat is provided for breaking up the water into spray, as shown between the buntons in the middle of the shaft. The water-flow is here conducted by a trough *W* into the spray-maker, where it is broken into drops and made to rain down the shaft in a rapid shower. This rain provides a powerful ventilation, and can be used with great advantage with a fall of from 100 to 200 feet. The arrangement is also cheap and economical where a copious mountain stream is available all the year round. So considerable is the pressure produced in this way, that waterfalls have been used to produce an air-blast for smelting iron in cupola furnaces. In some cases where water is available and can be used for ventilating mines, a vertical shaft to be used as a trompe is sunk on the side of a mountain at sufficient elevation above the top of the downcast. The bottom of the trompe is made a little below the top of the downcast shaft. The air from the fall is conducted into the downcast shaft by a drift, and to confine the air to the flow of the mine, the top of the shaft is covered with a trap-lid. The drift connecting the trompe and the downcast is extended past the downcast to the surface. The

FIG. 156.

water falling down the trompe is collected in a small sump, trapped into a trough in the drift, whence it flows past the downcast and runs away. Where the flow of water is copious, 200,000 or 300,000 cubic feet of air per minute can in this manner be supplied for the ventilation of a large mine.

VENTILATION BY STEAM-JET AND WATER-JET.

1014. Ventilation by Steam-Jet.—Sometimes a current of air is set in motion with a steam-jet projected into a channel along which the current moves, but economical results have not been obtained in this way.

1015. Ventilation by a Water-Jet.—Sometimes a jet of sprayed water is projected along the path of a current to produce ventilation. This method has not been very successful for producing large volumes of air, except when used as a waterfall, as previously described. But it has been applied with comparatively good results for producing a local current in a cheap and efficient way. It is only necessary, however, for the student to know that such means are used for setting currents of air in motion.

MECHANICAL VENTILATORS.

PRINCIPLES GOVERNING THE ACTION OF FANS.

1016. Chief among the mechanical ventilators of mines is the centrifugal fan, and it is, therefore, important that its principles of construction and mode of action should be understood. The fan is really a valveless pump, its blades taking the place of the pump-piston, and, so far as the exhausting and blowing out of the mine air is concerned, the fan and the pump act in the same way.

To set a fluid in motion, the ventilating fan, like the pump, must overcome three distinct causes of resistance; to make clear how these causes originate, an air-pump such

as shown in Fig. 157 is used. The first cause of resistance is that due to the friction of the mine. To show clearly its distinctive individuality, the pump is so contrived that no air can enter it without passing down the tube *A B* in the vessel at the left side of the figure; and to create an artificial resistance, the vessel just referred to is seen to be half filled with water, so that before any air can get inside of this closed vessel, two things must happen. First, the pump-piston *G* must move upwards, and, as a result of this movement, a depression of the air pressure will occur below the piston and above the water in the closed vessel. This

Fig. 157.

having taken place, the external air will by its greater pressure force the water down and out of the bottom end of the pipe *A B*, and it will then bubble up through the water and enter the pipe *D C* on its way into the cylinder, as shown by the arrow.

1017. The Depressions Produced by the Working of a Fan.—To make the depression required, the piston must move and produce a depression that will so reduce the pressure of the air in the cylinder and the vessel, that it will require the weight of the water the air displaces added to the pressure of the air within the vessel to equal the atmospheric pressure outside.

For example, suppose that the depth of the water through which the air must be forced is equal to 2 inches, or a pressure of 10.4 pounds per square foot. Then, if the outside pressure is equal to 2,116 pounds per square foot, the inside pressure can not be more than $2,116 - 10.4 = 2,105.6$ pounds. The piston has here made such a depression as is found in a fan drift, and that is the equivalent of what is called the mine resistance, or the pressure required to set a current of air in motion through a mine. The use of a fan is to make this depression. The first, or principal depression, and the equivalent of the force required to produce it, is represented in Fig. 157 by a weight MR hung on the opposite end of the beam that turns on the center pin S. The entire use of the vessel DLB is to generate an artificial resistance to imitate a mine resistance.

1018. The second cause of resistance is that due to the force required to set the air in motion in the cylinder through the port DC. Air can not be set in motion out of the vessel DLB without a depression in the cylinder at GC, for air-currents move only from a higher to a lower pressure. Therefore, the piston must move sufficiently to make a displacement not only equal to the depression the weight MR would produce, but, in addition, a depression equal to that which the weight IM would produce. This means that the depression within the cylinder at GC must be equal to $IM + MR = IR$, while the depression within the vessel DL will only be equal to MR. The depression IM is the one which represents the depression required for the entry of air into a fan.

1019. From what has been stated, the student can see that two depressions must be made by the action of the fan. The first one is provided to cause the fall of a current from the atmosphere through the mine into the fan drift, and the second one is provided for the air in the fan drift to fall into the fan. As has been shown, the sum of these depressions is equal to the pressure represented by the weights IM and

MR. In addition to these, however, there is a third resistance, OI, which is the pressure required for the air to fall out of the fan into the atmosphere. The piston can not force the air out of the upper end of the cylinder without a difference between the inside and the outside pressure; altogether, then, the sum of the pressures required for a fan to do its work is equal to OR, or $OI + IM + MR$.

1020. The centrifugal ventilating fan furnishes the best agent for the economical ventilation of mines, for two reasons: first, it is safer than the furnace; and, second, its efficiency is uniformly the same for deep and for shallow mines; whereas the efficiency of the furnace is very small for shallow mines, and is not much greater than the fan in the ventilation of deep ones. From all points of view, then, the centrifugal fan is the best ventilating machine in use.

COMPARISON OF FAN AND FURNACE.

1021. The underlying principles of the modes of action of the fan and the furnace are so different as to require particular notice. The ventilating pressure produced by the furnace is the result of the difference in the weights of the ventilating columns; whereas the ventilating pressure produced by a fan is the result of a difference in the total pressures upon two shafts. For example, if a pair of shafts are 1,200 feet in depth, and are ventilated by a furnace with the temperature of the downcast column 62° F., and that of the upcast 135° F., then by formula **57** the weight of a cubic foot of each can be found as follows: In the downcast, the weight of a cubic foot of air is equal to $\frac{1.3253 \times 30}{(459 + 62)} = .0763$ pound (see formula **57**), and the weight of a cubic foot of air in the upcast is equal to $\frac{1.3253 \times 30}{(459 + 135)} =$ .0669 pound. (See formula **57.**) The difference in the weights is, therefore, equal to $.0763 - .0669 = .0094$ pound. The pressure, per square foot, producing ventilation under

the given conditions of depth and heat, is, according to formula **60**,

$$\frac{(135 - 62)}{(459 + 135)} \times .0763 \times 1,200 = 11.252 \text{ pounds.}$$

1022. Fan ventilation is not produced, like furnace ventilation, by a difference in the weights of the ventilating columns. If, in the case of fan ventilation in the same shafts, the weight of a cubic foot of air in the downcast is equal to .0763 pound, the ventilating pressure of the fan is equal to that of the furnace, and the temperature of the upcast column is the same as that of the downcast one, namely, 62° F.; then, by taking the pressure of the atmosphere at 2,116 pounds per square foot, the weight of a cubic foot in the upcast shaft can be found by the following formula:

$$w = \frac{P - p}{P} \times W, \qquad \textbf{(62.)}$$

in which w = weight of 1 cubic foot of air in the upcast;
P = atmospheric pressure per square foot, or 2,116 pounds;
p = ventilating pressure in pounds per square foot;
W = weight of 1 cubic foot of air in the downcast.

Hence, in the case under consideration,

$$\frac{(2,116 - 11.252)}{2,116} \times .0763 = .075894 \text{ pound. Ans.}$$

The difference in the weight of a cubic foot of air in the downcast and of a cubic foot in the upcast is, therefore, equal to $.0763 - .075894 = .000406$ pound. That is, the weights of the upcast and downcast columns are practically the same.

To produce a ventilating pressure of 11.252 pounds per square foot with a furnace and with a fan, the following curious differences occur:

Differences in the weights of a cubic foot of air:

Furnace ventilation, .0094 pound.
Fan ventilation, .000406 pound.

Difference in the total pressures upon the ventilating columns, in the given examples:

For furnace ventilation, none; because the motion of the current is caused by a difference of weight in the two columns.

For fan ventilation, 11.252 pounds; direct pressure, with practically no difference in the weights of the two columns.

Plainly stated, the facts are these: The furnace rarefies the air by heat, and the air flows because the rarefied column is lighter than the other column. The fan, by exhaustion or compression, makes the total pressure upon the top of one column greater or less than the pressure upon the top of the other column; so that, although the weights of the two columns are practically the same, the difference in pressure on the tops of the columns produces the flow.

1023. From what has been explained, it is easy to infer that the mode of action that characterizes the centrifugal fan is that of producing differences of pressure between the air entering and leaving a mine, and entering and leaving a fan. For example, if the absolute pressure of the air within the fan drift were not below the external pressure of the air entering a mine, the air could not be set in motion. It is clear, then, that the prime object of the fan is to make a depression at one end of the mine so that the greater pressure at the other will set the air in motion. Further, after the air has passed through the passages of a mine and has reached the fan drift, it can not enter the fan unless the pressure within the fan is less than that of the air in the drift. Therefore, a fan, to produce a ventilating current, must make a provision for two distinct depressions, one to cause the air-current to flow towards the fan and one to cause the current of air to enter the fan itself. Again, air can not leave a fan unless its pressure is raised above that of the external air. Then, if the air leaving is at a pressure greater than that of the atmosphere, it is clear that the work to be done is equal to that of producing a motive pressure equal to the sum of two negative pressures and the positive

one. That is, it is necessary in this case to make a depression equal to about 10 pounds on the square foot to overcome the mine resistance, and a further depression of about 2 pounds on the square foot is required to cause the air to enter the fan. The sum of these depressions becomes $10 + 2$ pounds, or 12 pounds, on the square foot below the pressure of the atmosphere. Again, as the fan must make a compression for blowing the air out, say to 2 pounds on the square foot, the work of making the depression and the compression is altogether equal to $10 + 2 + 2 = 14$ pounds on the square foot.

1024. The above principles of action are explained by Fig. 158, in which a funnel and the pipe *A B* represent the

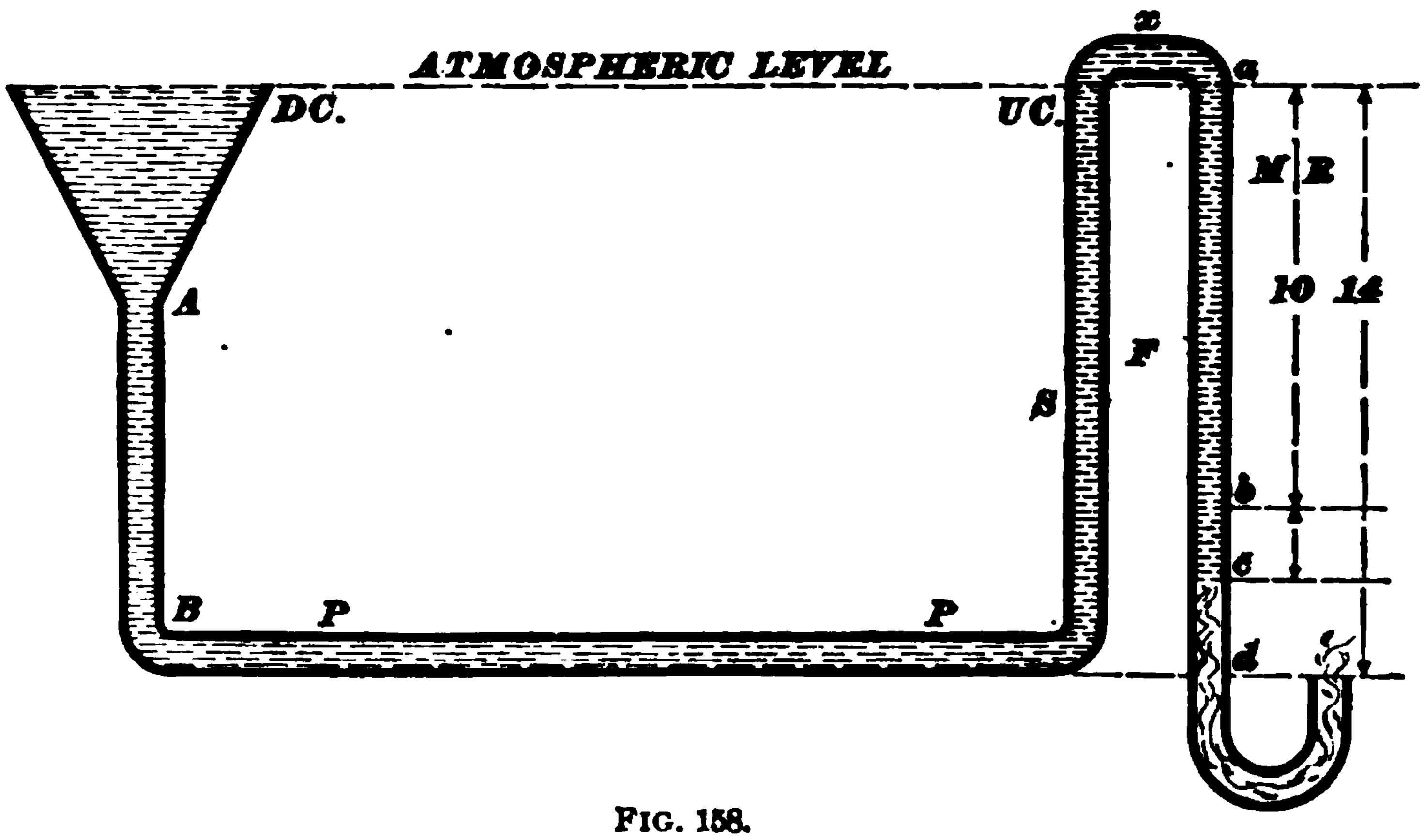


FIG. 158.

downcast shaft *D C*, and the pipe *S* forming one limb of the siphon represents the upcast shaft *U C*, while the descending limb *F* represents the depression and compression of a fan. It may be seen that, if at any moment the pressures or weights of the columns *D C* and *U C* are equal, the fluid will not flow through the pipes, because the atmospheric pressure will not force it through the elbow *x*; but as soon as a portion of it fills the descending limb *F*, a depression takes place in the column *S* and the fluid falls from *D C* to *U C*. Now, let us apply this principle to show

how an exhaust-fan produces ventilation. The horizontal pipe PP takes the place of the galleries in a mine, and the columns $A\,B$ and S take the place of the shafts, thus making $A\,B$ the downcast and S the upcast shaft. The falling limb of the siphon F represents the exhaust-fan. It now becomes important that, through the medium of this diagram, the two depressions and the single compression generated by the exhausting-fan be studied. In the first place, the fan represented by F makes a depression into which the air of the mine falls. If the mine resistance is nearly equal to 2 inches of water-gauge, or 10 pounds on the square foot, it can be graphically shown, as in that portion of the diagram at the right-hand side of the figure, that $a\,b$ is proportional to the depression required to overcome the mine resistance. Again, the depression required for the air to enter the fan is represented by $b\,c$; hence, for the air to flow through the mine and fall into the fan, a depression must be made equal to $a\,c$. Further, a pressure above the atmosphere is required to blow the air out of the fan, as shown at $c\,d$. The total amount of pressure then required to cause the air to flow through the mine and fall into the fan, and to blow it out, is $a\,b + b\,c + c\,d = a\,d$.

CALCULATION OF VELOCITIES AND PRESSURES.

1025. In calculating the resistance due to the flow of air through mine passages only, the well-known formula $p = \frac{k\,s\,v^2}{a}$ is used; but the pressure required to blow air into a fan, and blow it out, must be found in a different way, because the conditions that originate the resistance are different. For example, the greater portion of the resistance met by a current flowing through a mine is generated by the rubbing surfaces of the airways; but there are no rubbing surfaces to produce resistances when air moves through an orifice that practically has no length, as in the case of the port of entry into a fan and the port of discharge out of it.

There are resistances that are peculiar to orifices that have no length, such as the *vena contracta*. Now, but for this interference with the movements of fluids, air at a pressure of 2,116 pounds on the square foot would rush into a vacuum with a velocity whose square would be equal to 1,800,000. After this number has been corrected for the resistance due to the *vena contracta*, it is reduced from 1,800,000 to 685,600.

1026. The *vena contracta* is that resistance due to the divergent and convergent movements of the particles of a fluid moving through an orifice. To make this clear, a reference to Fig. 159 will show that the air particles *a*, *c*, *e*, *f*, and *h* are all converging towards *O*, the center of a fan orifice. As a consequence, their velocities in lines parallel to each other, and perpendicular to the plane of the orifice, are quite different; for example, in the time that the particle *e* requires to move to *O*, *c* moves to *d*, and *a* to *b*, or *f* to *g*, and *h* to *i*. Again, while *a* moves to *b*, the same particle is tending to move from *b* to *O*; since the latter movement is prevented, the direction of the particle is deflected, and this contracts the neck of the inverted cone of the inflow, and still further promotes the resistance at entry. Another cause of resistance is found in the whirl set up by the converging particles deflecting each other near the orifice, as at *D*, and this is the result of the velocities of the particles increasing as they accumulate in a reducing space, as *o p*, *q r*, or *n m*. These

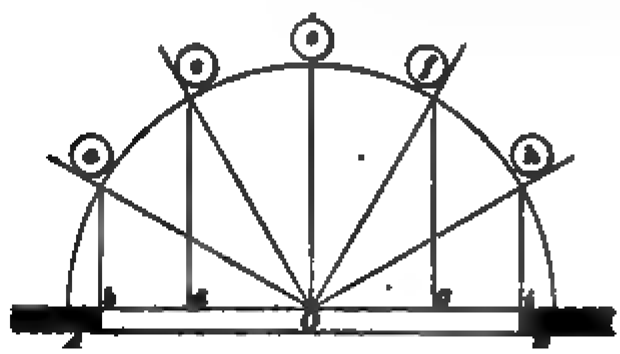


FIG. 159.

facts are still more clearly exemplified in Fig. 160. Here the lateral pressure has produced the contraction $D\ D$, and it is curious to observe that when the lateral pressure is relieved, the column swells out again to the full size of the orifice $A\ B$, and still further expands beyond the orifice, as at $E\ E$. Now, the result of these resistances is that the mean velocity of the inflow is reduced proportionately from 1 to .62;

FIG. 160.

for air this is called the coefficient of the inflowing velocity, or the *vena contracta*. It may thus be seen that when the particles of a current flow along converging lines, the channel of the stream is constricted, and the general velocity is reduced.

1027. If it requires an atmospheric pressure of 2,116 pounds per square foot to cause air to blow through an orifice into a vacuum with a mean velocity whose square is 685,600, then the square of the velocity for any other pressure is readily found, when it is remembered that the squares of the velocities of air-currents vary directly as the pressures. This principle may be stated by the following formula:

$$v = \sqrt{685{,}600 \times \frac{p}{2{,}116}}, \qquad \textbf{(63.)}$$

in which p = given pressure in pounds per square foot,
and v = velocity in feet per second.

Suppose the pressure is equal to 3 inches of water-gauge, or 15.6 pounds on the square foot; then, by formula **63**, $v = \sqrt{685{,}600 \times \frac{15.6}{2{,}116}} = 71.095$, the velocity in feet per second. It should be observed, however, that 685,600 and

2,116 are constant numbers, and that they can be eliminated by substituting a single constant.

Dividing 685,600 by 2,116, and extracting the square root of the quotient, formula **63** becomes

$$v = 18\sqrt{p}. \qquad \textbf{(64.)}$$

EXAMPLE.—Required the velocity with which air will move through an orifice under a pressure of 15.6 pounds per square foot.

SOLUTION.—Applying formula **64**,

$v = 18 \times \sqrt{15.6} = 71.095$, the velocity in feet per second. Ans.

The pressure required to blow air through an orifice when the mean velocity is given can be found by the formula

$$p = \left(\frac{v}{18}\right)^2. \qquad \textbf{(65.)}$$

EXAMPLE.—If air is blowing through an orifice, such as the entry into a fan, with a velocity of 71.095 feet per second, what will be the pressure or depression required ?

SOLUTION.—Applying formula **65**,

$p = \left(\frac{71.095}{18}\right)^2 = 15.6$ lb. per square foot. Ans.

The following equations show how formulas **64** and **65** are obtained:

$$\sqrt{685{,}600 \times \frac{p}{2{,}116}} = 18 \times \sqrt{p} = v,$$

$$\frac{v^2}{685{,}600} \times 2{,}116 = \left(\frac{v}{18}\right)^2 = p. \quad \text{Ans.}$$

1028. The following examples will illustrate the use of the formulas given:

EXAMPLE.—The mean velocity of the air blowing into the orifice of entry of a fan is 25.5 feet per second; what is the depression in pounds per square foot required to give this velocity ?

SOLUTION.—By formula **65**, the pressure will be equal to $\left(\frac{25.5}{18}\right)^2 =$ 2.006 lb. per sq. ft. Ans.

EXAMPLE.—The pressure by which air is blown into a fan is 2.006 pounds per square foot; what is the mean velocity of the entering air ?

SOLUTION.—By formula **64**, $v = 18\sqrt{2.006} = 25.5$ ft. per sec. Ans.

EXAMPLE.—The orifice for the entry of air into a fan is 10 feet in diameter, and the pressure of the external atmosphere is 2,116 pounds per square foot; where the air reaches its maximum depression within the fan, its pressure is equal to 2,104.5 pounds per square foot; and the pressure within the fan drift is equal to 2,106 pounds per square foot. (*a*) What is the pressure that is effective in blowing the air out of the drift into the fan? (*b*) What is the pressure in pounds per square foot that is equal to the mine resistance? (*c*) What is the quantity of air entering the fan in cubic feet per minute?

SOLUTION.—(*a*) If the maximum pressure within the fan is 2,104.5 pounds per square foot, and the maximum pressure in the fan drift is 2,106 pounds per square foot, the effective pressure blowing air into the fan is equal to $2{,}106 - 2{,}104.5 = 1.5$ lb. Ans.

(*b*) The pressure in pounds per square foot required to overcome the mine resistance is equal to $2{,}116 - 2{,}106 = 10$ lb. per sq. ft. Ans.

(*c*) The velocity of the air entering the fan in feet per second can be found by formula **64**; therefore, it is $18 \times \sqrt{1.5} = 22.0446$ feet per second. If the diameter of the orifice of entry is 10 feet, the area of the orifice in square feet will be equal to $10^2 \times .7854 = 78.54$ square feet; and as 60 times the velocity in feet per second is equal to the velocity in feet per minute, the following is the quantity of air entering the fan: $78.54 \times 22.0446 \times 60 = 103{,}883$ cu. ft. per min. Ans.

EXAMPLE.—The depression necessary for air to enter a fan is 1.5 pounds per square foot. The orifice of entry is 10 feet in diameter, and the area of the orifice of discharge is $\frac{3}{4}$ that of the orifice of entry. If, as has been shown, it requires 1.5 pounds per square foot to blow the air into the fan, what pressure per square foot will be required to blow it out?

SOLUTION.—The pressure to blow the air out will be inversely proportionate to the squares of the given areas. If the orifice of entry is 10 feet in diameter, then the area in square feet will be $10^2 \times .7854 = 78.54$ square feet, and the area of the orifice of discharge will be equal to $\frac{3}{4}$ of $78.54 = 58.905$ square feet; then the pressure required to blow the air out of the fan will be equal to

$$\left(\frac{78.54}{58.905}\right)^2 \times 1.5 = 2.666 \text{ lb. per sq. ft.} \quad \text{Ans.}$$

EXAMPLE.—The pressure required to overcome the frictional resistance of the air-currents in a mine is 12 pounds per square foot, and the quantity of air entering the fan is 150,000 cubic feet per minute. (*a*) What pressure is required to blow the air through an orifice of entry which is 12 feet in diameter? (*b*) What pressure will be required to blow the air out of the fan if the orifice of discharge has an area equal to $\frac{3}{4}$ of the area of the orifice of entry? (*c*) What will be the range of

pressure between the pressure of discharge and the maximum depression within the fan?

SOLUTION.—(*a*) To find the pressure required to blow the air into the fan, first find the velocity of the entering air in feet per second; that is, divide the quantity in feet per minute by 60 times the area of the orifice of entry, and the quotient will be the velocity in feet per second; thus, $v = \frac{150,000}{12^2 \times .7854 \times 60} = 22.1048$ feet per second. The pressure blowing the air into the fan is, by formula **65**, $p = \left(\frac{22.1048}{18}\right)^2 =$ 1.50809 lb. Ans.

(*b*) The area of the orifice of entry is $12^2 \times .7854 = 113.0976$ square feet, and as the orifice of discharge is $\frac{4}{5}$ of the area of the orifice of entry, it will be $113.0976 \times \frac{4}{5} = 90.478$ square feet.

The volume of air in cubic feet per minute leaving the fan is exactly the same as that entering it; if the areas of entry and discharge are different, the velocities must be inversely proportional to the areas, because the velocity must be greater through a small area than through a large one. In this example the velocity through the large area is to that of the small one as 1 is to $\frac{113.0976}{90.478}$. Again, the pressures vary as the squares of the velocities, and, therefore, the pressure required to blow the air out is

$$\left(\frac{113.0976}{90.478}\right)^2 \times 1.50809 = 2.35639 \text{ lb. per sq. ft. Ans.}$$

(*c*) The total range of pressure between the pressure of discharge and the maximum depression within the fan can be found as follows:

Mine resistance	12.00000 pounds.	
Blowing-in pressure.........	1.50809 pounds.	
Blowing-out pressure........	2.35639 pounds.	
Total	15.86448 pounds.	Ans.

EXAMPLE.—If a pressure of 2 pounds per square foot is required to blow air out of a fan which has an orifice of discharge equal to 95 square feet, what depression will be required to blow air into the same fan when the orifice of entry has an area of 120 square feet?

SOLUTION.—As in the above example, the required pressure will be the ratio of the squares of the given areas multiplied by the given pressure; or $p = \left(\frac{95}{120}\right)^2 \times 2 = 1.253472$ lb. per sq. ft., the pressure required to blow air into the fan. Ans.

EXAMPLES FOR PRACTICE.

1. The velocity of air blowing through an orifice is 45 feet per second; what pressure per square foot is required to give this velocity?
Ans. 6.25 lb. per sq. ft.

2. When the pressure required to blow air through an orifice is equal to 5 pounds per square foot, what velocity will be produced?
Ans. 40.2492 ft. per sec.

3. The velocity of air blowing through an orifice is equal to 180 feet per second; what pressure per square foot will be required to give this velocity? Ans. 100 lb. per sq. ft.

4. With what velocity can air be blown through an orifice under a pressure of 120 pounds per square foot? Ans. 197.18 ft. per sec.

5. If it requires a depression of 1.5 pounds per square foot for air to blow into the port of entry of a fan that is 12 feet in diameter, what pressure would be required to blow the air out through a port of discharge that is 10 feet in diameter? Ans. 3.1104 lb.

6. If it requires a pressure of 4 pounds per square foot to blow air through the port of discharge of a fan that has an area of 90 square feet, what pressure will be required for air to enter the same fan when the port of entry has an area of 150 square feet?
Ans. 1.44 lb. per sq. ft.

7. What will be the total pressure of the air just within the port of entry of a fan when the atmospheric pressure is 2,116 pounds per square foot, the mine resistance is equal to 10.4 pounds per square foot, and the depression necessary for the air to blow into the fan is 1.2 pounds per square foot? Ans. 2,104.4 lb. per sq. ft.

1029. The total range of pressure by which a ventilating fan does its work extends from the maximum depression within the fan to the maximum compression without it

For example, suppose the following are the totals of the pressures:

Mine resistance.........	10.0 pounds.
Blowing-in pressure.....	1.5 pounds.
Blowing-out pressure ...	2.0 pounds.
Total range of pressure..	13.5 pounds.

The limits of the total range of pressure arise in this way: If the total pressure of the external atmosphere is 2,116 pounds per square foot, this constitutes an actual depression into which the air from the fan is blown; consequently, the maximum pressure in this example is $2,116 + 2 = 2,118$, and

the minimum pressure within the fan is $2{,}118 - 13.5 = 2{,}104.5$ pounds per square foot. Now, $2{,}118 - 2{,}104.5 = 13.5$ pounds, as previously shown.

EXAMPLE.—What pressure per square foot will be required to blow 150,000 cubic feet of air per minute into a fan (*a*) when the orifice of entry is equal to 10 feet in diameter, and (*b*) when the orifice of entry is equal to 5 feet in diameter?

SOLUTION.—(*a*) The velocity in feet per second of the air passing through the orifice 10 feet in diameter is found as follows:

$$\frac{150{,}000}{10^2 \times .7854 \times 60} = 31.831 \text{ feet per second. By formula } \mathbf{65},$$

$$p = \left(\frac{31.831}{18}\right)^2 = 3.1272 \text{ lb. per sq. ft. Ans.}$$

(*b*) In the same manner the velocity of the air entering the orifice 5 feet in diameter is found to be $\frac{150{,}000}{5^2 \times .7854 \times 60} = 127.323$ feet per second, and the required pressure, by formula **65**, is equal to $\left(\frac{127.323}{18}\right)^2 = 50.035$ lb. per sq. ft., the pressure per square foot required to blow 150,000 cubic feet of air per minute through an orifice 5 feet in diameter. Ans.

The pressure required to blow 150,000 cubic feet of air per minute through an orifice 5 feet in diameter is 16 times greater than the pressure required to blow air through an orifice 10 feet in diameter.

For, to blow equal quantities through unequal areas in equal times, the pressures vary inversely as the fourth powers of the diameters of the orifices. To prove the statement, let the quantity be 150,000 cubic feet of air per minute, and let the pressure for an orifice 10 feet in diameter be 3.1272 pounds per square foot; then the pressure per square foot required to blow the same volume of air per minute through an orifice 5 feet in diameter is equal to $\left(\frac{10}{5}\right)^4 \times 3.1272 = 50.035$ pounds, as in the above example.

DIMENSIONS OF THE PORTS OF A VENTILATING FAN.

1030. To obtain the best results with the ventilating fan, the depression necessary for the entry of air should, if possible, not exceed one pound per square foot. Hence, the velocity should not exceed 18 feet per second; for, by formula

64, $v = 18 \times \sqrt{p}$, and $18 \times \sqrt{1} = 18$. Now, 18 feet per second is equal to $18 \times 60 = 1{,}080$ feet per minute. Using this velocity, the diameter of the port of entry may be found by the following formula:

$$d = .0343 \sqrt{q}, \qquad \textbf{(66.)}$$

where d is the diameter of the port of entry and q is the quantity of air flowing per minute through *one* port of entry. If there are two ports, that is, if the fan receives air on both sides, q is obtained by dividing the total quantity per minute by 2.

EXAMPLE.—What should be the theoretical diameter of the port of entry of a fan to pass 200,000 cubic feet of air per minute?

SOLUTION.—Using formula **66**,

$$d = .0343 \sqrt{q} = .0343 \sqrt{200{,}000} = 15.355 \text{ ft. Ans.}$$

1031. The area of the throat of a fan must be equal to the area of the curved surface of an imaginary cylinder whose diameter is equal to that of the port or ports of entry; and, as the length of this cylinder is exactly equal to the breadth of the fan-blades, it is important that the relationship of this area to that of the port of entry should be fully understood. The breadth of the blades or the length of the imaginary cylinder just referred to is found as follows:

Let $d =$ diameter of port of entry;
$b =$ breadth of blades.

Then the curved surface of the imaginary cylinder is

$$3.1416\, d\, b$$

the area of the port of entry is

$$.7854\, d^2$$

Therefore, $3.1416\, d\, b = .7854\, d^2$;

or, $$b = \tfrac{1}{4} d. \qquad \textbf{(67.)}$$

This formula is applied when there is but one port of entry. When there are two ports of entry, $b = \frac{1}{2} d$.

EXAMPLE.—What should be the width of blade of a fan which is to deliver 160,000 cu. ft. of air per minute, there being one port of entry?

SOLUTION.—Using formula **66**,

$$d = .0343\ \sqrt{160,000} = 13.72 \text{ ft.}$$

Now, applying formula **67**,

$$b = \tfrac{1}{4} d = \tfrac{1}{4} \times 13.72 = 3.43 \text{ ft. Ans.}$$

EXAMPLE.—If 160,000 cubic feet are delivered per minute and there are two ports of entry, what should be the diameter of each port of entry and the width of the blade?

SOLUTION.— $q = \dfrac{160,000}{2} = 80,000$. Using formula **66**,

$$d = .0343\ \sqrt{80,000} = 9.7 \text{ ft. Ans.}$$

$$b = \tfrac{1}{2} d = 4.85 \text{ ft. Ans.}$$

The area of the port of discharge in an ideal fan should not be less than .81 of the area of the port of entry. It is true that few fans will work satisfactorily when this port is so large, but such fans can not give best results, because when the area of the discharge port is too much restricted, the excessive pressure required to blow out the air is much greater than it should be. Again, if there is not enough constriction in the port of discharge, there is bound to be excessive vibration of the air in the fan, necessitating the employment of a shutter. Therefore, .81 is far above the average proportion in many fans, but it is an ideal that should be sought for.

EXAMPLE.—The area of the port of entry of a fan is equal to 150 square feet, and the area of the port of discharge is .6 of this; then, if a pressure of 1 pound per square foot is required to blow the air into the fan, what pressure will be required to blow it out?

SOLUTION.—The pressures for blowing in and blowing out are inversely proportional to the squares of the areas; therefore,

$$\left(\frac{150}{150 \times .6}\right)^2 \times 1 = 2.77 \text{ lb. per sq. ft. Ans.}$$

EXAMPLE.—If the area of the port of entry of a ventilating fan is equal to 150 square feet, and the area of the port of discharge is equal to .81 of the area of port of entry, and if a pressure of 1 pound per square foot is required to blow the air into the fan, what pressure will be required to blow it out?

SOLUTION.—The area of the port of discharge will be $150 \times .81 = 121.5$ square feet, and the pressure to blow the air out of the fan will be

$$\left(\frac{150}{121.5}\right)^2 \times 1 = 1.52415 \text{ lb. per sq. ft. Ans.}$$

EXAMPLE.—The area of the port of entry of a ventilating fan is 150 square feet, and the area of the port of discharge is .5 of the area of the port of entry. If it requires 1 pound of depression to blow the air in, what compression will be required to blow it out?

SOLUTION.—The area of the port of discharge will be $150 \times .5 = 75$ square feet, and the pressure to blow the air out will, therefore, be

$$\left(\tfrac{150}{75}\right)^2 \times 1 = 4 \text{ lb. per sq. ft. Ans.}$$

THE MANOMETRIC EFFICIENCY.

1032. Manometric efficiency is that percentage of the total pressure generated by a ventilating fan that is efficient in blowing the ventilating current through a mine. What is here called the total pressure consists of three additive quantities:

1. The mine resistance M in pounds per square foot, as measured with the water-gauge.
2. The depression I required for the air to enter a fan.
3. The pressure O required to blow the air out of a fan.

Let A = area of port of entry;
a = area of port of discharge;
C = manometric efficiency.

Then the pressure O is given by the following formula:

$$O = \frac{A^2}{a^2} \times I. \qquad \textbf{(68.)}$$

The percentage of manometric efficiency C is found by formula **67**, where

$$C = \frac{100\,M}{M + I + O}. \qquad \textbf{(69.)}$$

EXAMPLE.—What is the manometric efficiency of a ventilating fan when the mine resistance is 2.5 inches of water-gauge, the depression at the port of entry of the fan is 2 pounds per square foot, the area of the port of entry is 100 square feet, and the area of the port of discharge is 60 square feet?

SOLUTION.—By formula **68**, $O = \frac{100^2}{60^2} \times 2 = 5.555$ lb.

Again, by formula **69**, $C = \frac{100 \times 13}{(13 + 2 + 5.555)} = 63$ per cent. Ans.

EXAMPLE.—Required the percentage of useful effect or manometric efficiency of a ventilating fan when the mine resistance is equal to 12 pounds per square foot, the depression required for blowing air into the fan is equal to 1 pound per square foot, and the compression for blowing the air out is equal to 1.5 pounds per square foot?

SOLUTION.—The total range of pressure is:

Mine resistance	12.0 pounds.
Blowing-in pressure	1.0 pound.
Blowing-out pressure	1.5 pounds.
Total	14.5 pounds.

The pressure required for mine resistance alone is 12 pounds; therefore, the efficiency of the fan, in so far as the ventilating of the mine is concerned, is $\frac{12}{14.5} \times 100 = 82.7586$ per cent. Ans.

The last two examples are given to show the importance of making the ports of entry and discharge sufficiently large to prevent needless waste in the working of a ventilating fan.

EXAMPLE.—The mine resistance is equal to 10 pounds per square foot, the blowing-in pressure is equal to 2 pounds per square foot, and the area of the port of discharge is so small that the blowing-out pressure is 8 pounds per square foot; what is the percentage of useful effect, or manometric efficiency of the fan as a ventilator?

SOLUTION.—The total range of pressure is equal to the following sum:

Mine resistance	10 pounds.
Blowing-in pressure	2 pounds.
Blowing-out pressure	8 pounds.
Total	20 pounds.

Therefore, the efficiency of the fan as a ventilator is equal to $\frac{10}{20} \times 100 =$ 50 per cent. Ans.

EXAMPLE.—A fan is exhausting from a mine 180,000 cubic feet of air per minute, and the area of the port of intake is 60 square feet; what is the pressure required for blowing the air into the fan?

Solution.—First find the mean velocity of the entering air in feet per second as follows:

$$\frac{180,000}{60 \times 60} = 50 \text{ feet per second;}$$

therefore, by formula **65**, the required pressure is $(\frac{50}{18})^2 = 7.716$ lb. per sq. ft. Ans.

EXAMPLES FOR PRACTICE.

1. What is the manometric efficiency of a fan when the mine resistance is equal to 15 pounds per square foot, the blowing-in pressure is 2 pounds per square foot, and the blowing-out pressure is 5 pounds per square foot? Ans. 68.18 per cent.

2. The manometric efficiency of a fan is 70 per cent., and the mine resistance is 3 inches of water-gauge; what are (*a*) the blowing-in and (*b*) the blowing-out pressures when the area of the port of discharge is .6 of the area of the port of entry? Ans. $\begin{cases} (a)\ 1.7672 \text{ lb. per sq. ft.} \\ (b)\ 4.918 \text{ lb. per sq. ft.} \end{cases}$

CENTRIFUGAL FORCE.

1033. The tendency of every body in motion is to move in a straight line, unless the body is acted upon by some force which causes it to deviate from the straight line. In the case of a body attached to a string and moving in a circle, the deviating force is the pull exerted by the string, and is called **centripetal force.** The so-called **centrifugal force** is that force which is equal and opposite to the centripetal force; in other words, it is a reaction. Centrifugal force can not cause motion; hence, if the string were cut, the centripetal force would no longer act, and the body, being then free to move, would start off in the direction of a straight line tangent to the circle, as shown by the line $e\,g$ in (*a*), Fig. 161. Centrifugal force is manifested under two conditions. In the first case there is a uniform deflection and a uniform velocity with a constant radius, as, for instance, when a body is made to rotate with a uniform velocity at the outer extremity of a constant radius. The centrifugal force can be found by the formula

$$f = \frac{w\,v^2}{R\,g}, \qquad \textbf{(70.)}$$

in which w = weight in pounds;
v = velocity in feet per second;
R = radius in feet;
g = acceleration due to gravity, or 32.16;
and f = force in pounds pulling the body towards the center of its revolution.

This is a case like that in which a pound weight might be made to revolve on the end of a string, and should the string at any moment be cut, the weight would simply move off in a line tangent to the curve.

In the second case, the centrifugal force, the velocity, and the radius are constantly increased, and correspond to the centrifugal force developed by a ventilating fan, which develops a velocity outwards from the center of revolution. To explain the difference, take a case in which a pound weight is made to revolve within a tube instead of being attached to the end of a string. Now, if the tube moves with the same speed as the string, the pound weight will move outwards along the inside of the tube. The velocity thus acquired will cause the weight to move in a path situated between the tangent to the curve and the prolongation of the radius at the point of disengagement, instead of in a path tangential to the curve of the outer circle. The body revolving on the end of the fixed radius has, at the moment of disengagement, acquired only sufficient centrifugal force to make it describe a path tangential to its circle; whereas, the body moving through the tube has, in addition to the centrifugal force of the former body, acquired a force due to an increased outward acceleration.

The force acquired in the second case is calculated by the formula

$$f = \frac{w v^2}{3.1416 g}, \qquad \textbf{(71.)}$$

in which the factors are the same as in formula **70**, except that the constant 3.1416 is substituted for R.

1034. In (a), Fig. 161, both of the above cases are illustrated. The radius of the circle described by the first

body revolving is *o e*, or *o h*, and if this body is by any means disengaged, as, for example, by the breaking of a string, at the instant it is passing the point *e*, the body will reach the point *g* at the moment it should have arrived at *h*, having moved along the line *e g*, which is tangent to the curve *e h*. If, however, the same body is made to revolve in a tube and also to commence its outward journey at the center *o*, by the time it reaches *e* it will have acquired a high outward velocity that the fixed body can not possess. Therefore, at the moment of disengagement it will move off in the path *e f*, and will arrive at *f* in as short a time as the

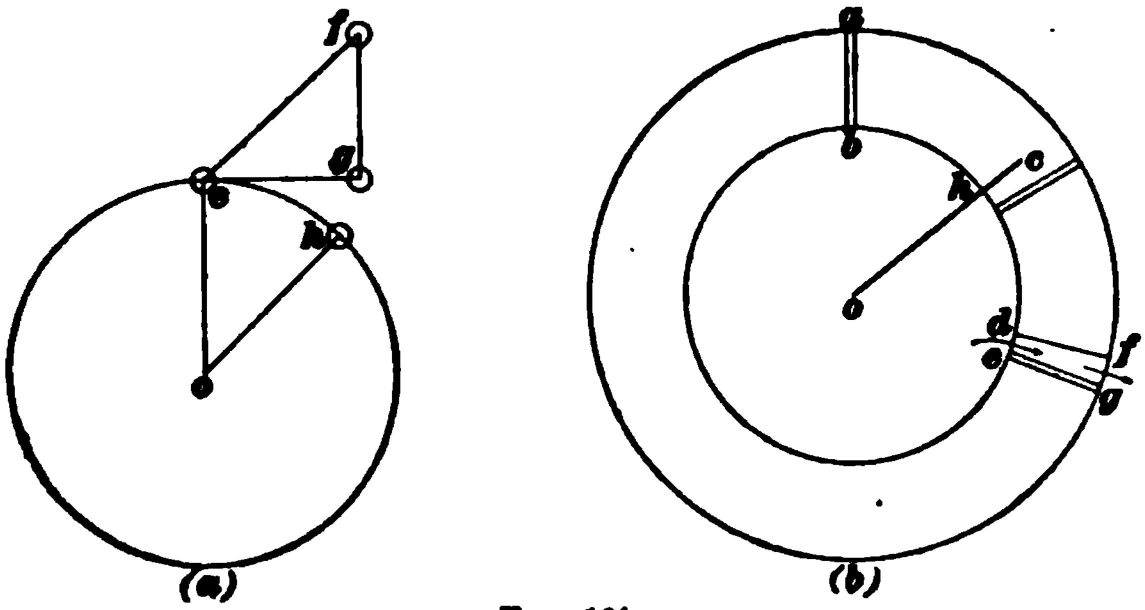


FIG. 161.

first body will require to reach *g*. The result is that the velocity developed by a continual deflection and acceleration is much greater in amount than that due to a body revolving with a uniform velocity on the outer extremity of a radius of constant length.

DEFINITIONS OF TERMS.

1035. The centrifugal force developed by a fan is calculated by the following formula :

$$f = \frac{w\,l\,v^2}{3.1416\,g}. \qquad (72.)$$

Before giving examples that involve the use of formula **72**, the meaning and use of its factors must be explained. For example, the velocity is obtained from the length of the radius of gyration, and this is obtained by adding to the radius of the port of entry a fraction of the radial length of

the blades, the latter being found by multiplying the radial length by .6. The radial length of the blades is found by the formula

$$l = \frac{D - d}{2}, \qquad (73.)$$

in which D = diameter of the fan;
d = diameter of the port of entry;
and l = radial length of the blades.

EXAMPLE.—The diameter of a fan is 30 feet, and the diameter of the port of entry is 12 feet. What is the radial length of the blade?

SOLUTION.—By formula **73**, the radial length of the blades is $\frac{30 - 12}{2} = 9$ ft. Ans.

1036. In (*b*), Fig. 161, is given a graphic illustration of the terms in question. For example, taking o as the center of a fan, c is situated at the center of gyration, and $o\,c$ is the radius of gyration. As $o\,h$ is the radius of the port of entry, $o\,h + h\,c$ (or, what is the same thing, $o\,h + a\,b \times .6$) is equal to the radius of gyration. This is expressed by the formula

$$r = \frac{d}{2} + .6\,l, \qquad (74.)$$

in which r is the radius of gyration. When air is flowing along the blades of a fan, all the air on the blades, from the periphery of the port of entry to the outer periphery of the fan itself, is subject to centrifugal force, and, as the blades may be 9 or 10 feet long, to find the weight of air subject to centrifugal force, the weight of a cubic foot of air is multiplied by l, the length of the blades; this is the meaning of the expression in formula **72**, where $w\,l$ occurs as two of the factors.

Now, it must be clear that the velocity of the moving air at b is much less than it is at a; and, therefore, the mean of the squares of the velocities, multiplied by the weights, occurs at the center of gyration c, for a stream of air lies on every blade, as that shown at $d\,e\,g\,f$. An example will make this clear:

EXAMPLE.—A ventilating fan is 30 feet in diameter, the radial length of the blades is 9 feet, and the length of the radius of gyration is 11.4 feet; (*a*) what is the mean velocity generating centrifugal force when the fan is making 50 revolutions per minute? (*b*) What is the total pressure produced? (*c*) What is the quantity of air passed per minute?

SOLUTION.—The velocity generating centrifugal force in feet per second will be equal to $\frac{11.4 \times 2 \times 3.1416 \times 50}{60} = 59.69$ feet per second, the required velocity. From this velocity, the total pressure in pounds per square foot to produce the two depressions already noticed and the compression for blowing out can be found by formula **72.** In the case for which the velocity has been calculated, the length of the blade is 9 feet; if the average weight of a cubic foot of air is taken at .076 pound, formula **72** gives $f = \frac{.076 \times 9 \times 59.69^2}{3.1416 \times 32.16} = 24.11$ pounds per square foot. Next, suppose that the mine resistance in a case like this is equal to 3 inches of water-gauge, or 15.6 pounds per square foot, and the depression within the fan is 4 pounds per square foot below that in the fan drift, and that the pressure per square foot for blowing out is 4.51 pounds above the atmosphere. These figures yield the factors for calculating the quantity of air that this fan is exhausting out of the mine in cubic feet per minute. By formula **64,** $v = 18\sqrt{p} = 18 \times \sqrt{4} = 36$ feet per second, the mean velocity of the air entering the fan. Next, the port of entry, which is circular, is $30 - 2l$, or $30 - 18 = 12$ feet in diameter, and its area is $12^2 \times .7854 = 113.0976$ square feet. 36 is the mean velocity in feet per second, and, therefore, $36 \times 60 = 2{,}160$, the velocity in feet per minute. If the area found be multiplied by the mean velocity in feet per minute, the result will be the quantity of air exhausted by the fan in cubic feet per minute, as $113.0976 \times 2{,}160 = 244{,}290.8$ cubic feet of air per minute. Ans.

EXAMPLE.—A ventilating fan is 28 feet in diameter, and the diameter of the port of entry is 10 feet; what is the radial length of the blades?

SOLUTION.—By formula **73,**

$l = \frac{D - d}{2}$; then, $\frac{28 - 10}{2} = 9$ feet, the radial length of the blades. Ans.

EXAMPLE.—A ventilating fan is 28 feet in diameter, and the orifice of entry is 10 feet in diameter; what is the length of the radius of gyration?

SOLUTION.—By formula **74,**

$$r = \frac{d}{2} + .6 \times l;$$

or, $r = \frac{10}{2} + (.6 \times 9) = 10.4$ ft., the radius of gyration. Ans.

EXAMPLE.—The radius of gyration for a ventilating fan is 9.5 feet, and the length of blade is 7.5 feet; the diameter of the orifice of entry is 10 feet; the angular velocity is 50 revolutions per minute; what is the total range of the fan's ventilating pressure?

SOLUTION.—The velocity per second is equal to $\frac{9.5 \times 2 \times 3.1416 \times 50}{60} =$ 49.742 feet; then, by formula **72**,

$$f = \frac{w\,l\,v^2}{3.1416\,g} = \frac{.076 \times 7.5 \times 49.742^2}{3.1416 \times 32.16} = 13.96 \text{ lb.},$$

the total ventilating pressure. Ans.

1037. The Center of Gyration.—The calculations thus far given are based on the assumption that the blades of the fan lie longitudinally in radial lines, but in many cases this does not occur. Therefore, it is necessary to be able to make the expressions adaptable for fans in which blades make different angles with the radii. Now, when the blade makes an angle with the radius, its efficient length is practically shortened in the proportion of the cosine of the angle. For example, suppose a blade makes an angle of 45° with the radius projected from the circumference of the port of entry; then the efficient length of the blade is only .7 of its actual length, as shown in Fig. 162, in which $A\,B$ is the radius, $C\,D$ the actual length of the blade, and $C\,B$ the efficient length of the blade. Again, when the blades make an angle with the radii, the efficient angular velocity, that is, the number of revolutions, is practically reduced in the proportion of the cosine of the angle which the blades make with the radii. For, suppose a case in which the fan is making 50 revolutions per minute. The 50 would be reduced by the cosine of the angle, because where the blades decline from the radii, the relative efficiency of the velocity is only for an angle of 45°, which is equal to $50 \times .7 = 35$ revolutions per minute. To make this allowance, the expressions may be simplified by multiplying the number of revolutions by the square of the cosine; formula **72** thus becomes

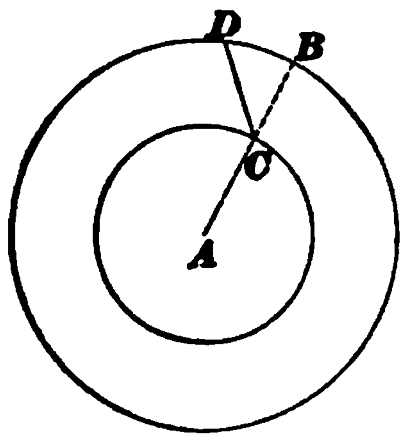


FIG. 162.

$$f = \frac{w\,l\,v^2\,(\cos a)^2}{3.1416\,g}, \qquad (75.)$$

in which a = angle the blades make with the radii in every case;
f = total pressure per square foot;
v = velocity of the center of gyration in feet per second;
l = actual length of the fan blades;
w = weight of a cubic foot of air, or .076;
g = acceleration due to gravity, or 32.16;
and 3.1416 = a constant.

If, for example, the blades of a fan are each 8 feet long, and are so set as to make an angle of 30° with the radii, and the velocity of the center of gyration is 60 feet per second, the total pressure, by use of formula **75,** is calculated as follows:

$$f = \frac{3{,}600 \times .076 \times 8 \times .86603^2}{3.1416 \times 32.16} = 16.25 \text{ lb., total pressure.}$$

1038. The importance of being able to calculate the mean velocity of a current of air flowing outwards along the blades of a fan can not be overlooked, because, when this velocity is not known, the formula adopted only secures a rough approximation of what a ventilating fan is capable of doing. To understand the matter, the student must first have clear views concerning how air moves in its passage through a fan, and the best way of obtaining this is to consider in order the three components of the resultant motion.

1039. *First.*—The angular velocity is that due to the revolution of the fan. As all the particles in a current of air flowing along the blades of a fan make a revolution round the common center of motion in the same time, it is necessary to explain how the angular velocity affects the sum of the work of a fan. Angular velocity means the same thing as revolutions per minute or per second, for the angular velocity increases or decreases directly as the number of revolutions increases or decreases.

1040. *Second.*—While all the particles of air within a fan have the same angular velocity, the linear velocities are directly proportional to their radial distances from the center of motion; for example, all the particles marked *6*, *7*, *8*, *9*, *10*, *11*, *12*, in the upper portion of Fig. 163, have the same angular velocity; that is, they all make one revolution in the same time, but their linear velocities are different, for they are all describing circles of different lengths. The linear velocity of *6* is proportional to the length of the arc *a b*. The linear velocity of *12* is proportional to the length of the arc *m n*, and the same relationship holds true with all the other numbers.

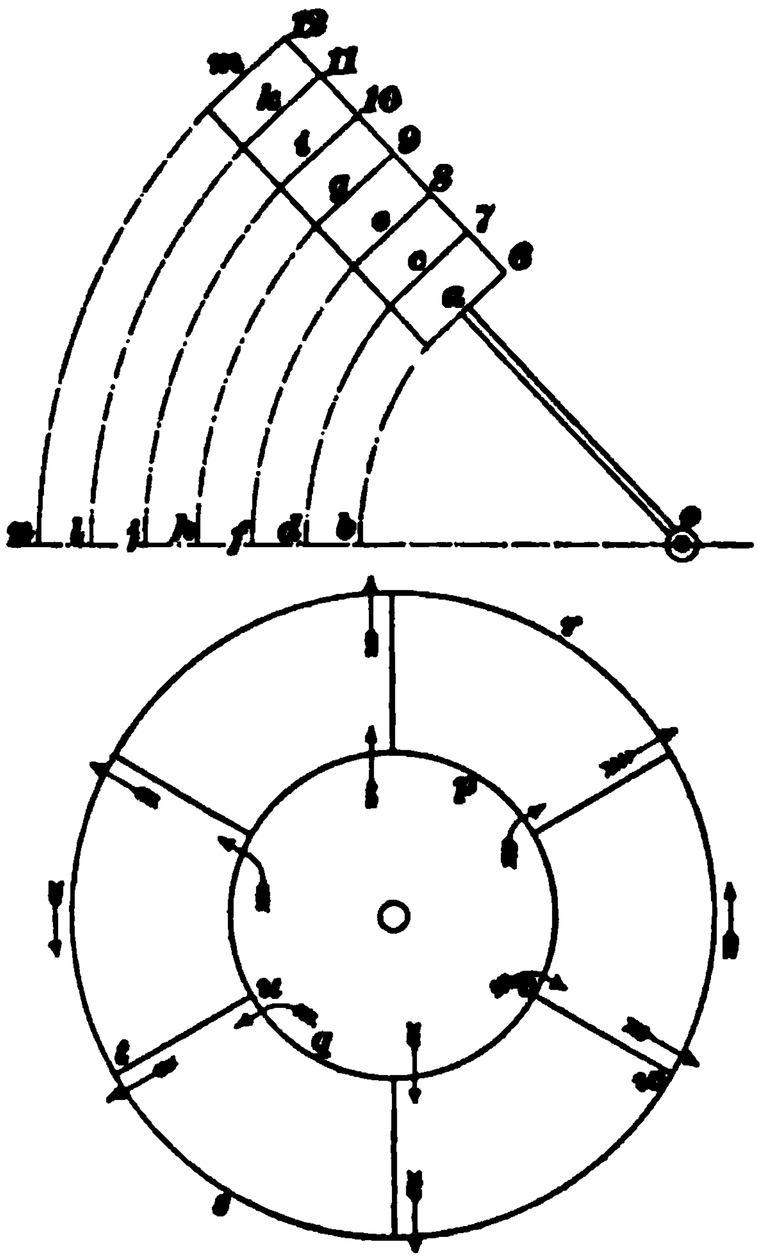


FIG. 163.

1041. *Third.*—What is called radial motion is the flow of the stream of air along the blades of a fan. There are many mistaken ideas in regard to this motion. For example, the common idea is that the currents of air entering a fan diffuse between the diverging blades, and, therefore, the velocity reduces as the divergence increases. A little consideration, however, shows the error of this conclusion. The air-currents, as a whole, flow through a fan with an unvarying velocity, for the weight of the air leaving a fan can never exceed the weight of the air entering it, but the particles making up the currents have different velocities in the direction of the fan's motion. Again, the increasing effect of the centrifugal force, as the stream advances to the circumference of the fan, tends more and more to prevent

any increase in the depth of the flow; indeed, it tends rather to reduce it. This is fully proved on watching the contraction that takes place in a river where it contracts its flow as the result of the effect of the centrifugal force that is generated by a bend in the channel. The flow of the air is such as indicated by the arrows along the blade of the fan, as seen in the lower portion of Fig. 163. The fan is supposed to be turning in the direction of the hands of a watch, and the inner circular space within the fan blade is the port of entry, or the orifice by which all air enters a fan.

1042. The Evase Chimney.—Fig. 164 is an illustration of four important points. The first is the fan drift illustrated by the round tube *I*; the second is the fan-drift depression, as shown by the water-gauge *D*; the third is the water-gauge *G*, that shows the depression that occurs within the fan; and the fourth is the greater pressure or compression of the air at discharge, as shown by the water-gauge *P*. These points have all been explained. *E* is the evase chimney used to reduce the amount of waste that occurs in blowing air out of a fan into the external atmosphere. In the case of the Guibal fan, for instance, where the orifice of entry is relatively small and the velocity of discharge is relatively high, if something is not done to reduce the high velocity, the blast of air striking the comparatively still air of the atmosphere causes a rebound that produces great resistance. The evase chimney is used to reduce this needless loss of energy in the discharged air. To show the efficiency of an evase chimney, assume the orifice of discharge from the casing to the chimney

Fig. 164.

to be equal to 50 square feet, and the area of the top of the evase chimney to be equal to 200 square feet. Now, the velocity of discharge at the top of the evase chimney will only be one-fourth of the velocity of the air blowing through the orifice of discharge of the fan, and as the resistances that arise when air at a high velocity strikes still air are proportionate to the squares of the velocities, the resistance due to the air leaving the top of the evase chimney is to the resistance of the higher velocity as 50^2 is to 200^2 or as 1 is to 16.

EXAMPLES FOR PRACTICE.

1. What is the centrifugal force in pounds due to a 5-pound weight revolving on the end of a rigid radius under the following conditions: Length of radius, 6 feet; number of revolutions per second, 10?
Ans. 3,682.7 lb.

2. What is the pressure per square foot due to a stream of water flowing through a pipe that is rotating on one of its ends, the length of the pipe being 5 feet and making 2 revolutions per second?
Ans. 4,395.9 lb. pressure per sq. ft.

3. A ventilating fan is 25 feet in diameter, and the port of entry is 10 feet in diameter. What is the radial length of the blades?
Ans. 7.5 ft.

4. A fan is 25 feet in diameter, and the diameter of the port of entry is 10 feet. What is the length of the radius of gyration?
Ans. 9.5 ft.

5. A ventilating fan is 25 feet in diameter, the port of entry is 10 feet in diameter, and the radius of gyration is 9.5 feet. What is the velocity in feet per second of the center of gyration when the fan makes 45 revolutions per minute? Ans. 44.7678 ft. per sec.

6. The velocity of the center of gyration of a ventilating fan is 44.7678 feet per second, and the radial length of the blades is 7.5 feet. What is the total pressure generated by the fan to overcome the mine resistance, and to set the air in motion into and out of itself?
Ans. 11.3032 lb. per sq. ft.

TYPES OF FANS.

1043. Centrifugal fans are used for blowing and exhausting. Exhausting fans are in most general use, though there are many advocates and users of the blowing-fan. So far as mechanical efficiency is concerned, exhaust-fans and blowing-fans are practically equal.

The general principles of each are the same, except in reverse order. The exhaust-fan draws from the mine and discharges into the outer atmosphere, while the blowing-fan draws from the outer atmosphere and discharges into the mine.

1044. Prominent Types of Fans.—The most prominent types of centrifugal ventilators now in use in mining countries are four in number, the principal representatives of which are (*a*) *Waddle*, (*b*) *Schiele*, (*c*) *Guibal*, (*d*) *Capell*. These four will be described as representing the main features of all other forms, which are modifications of these original types.

1045. The Waddle Fan.—The characteristic features of the *Waddle* fan, Fig. 165, are the *curvatures of its blades*—backwards from the direction of their motion; their tapered form, tapering towards the circumference, and the tight box sides, which revolve with the blades. The blades leave the orifice of intake radially, but curve backwards from the direction of their motion till they are almost tangential at the circumference. The blades are so tapered from the orifice of intake to the circumference that the breadths of the blades at

FIG. 165.

to be equal to 50 square feet, and the area of the top of the evase chimney to be equal to 200 square feet. Now, the velocity of discharge at the top of the evase chimney will only be one-fourth of the velocity of the air blowing through the orifice of discharge of the fan, and as the resistances that arise when air at a high velocity strikes still air are proportionate to the squares of the velocities, the resistance due to the air leaving the top of the evase chimney is to the resistance of the higher velocity as 50^2 is to 200^2 or as 1 is to 16.

EXAMPLES FOR PRACTICE.

1. What is the centrifugal force in pounds due to a 5-pound weight revolving on the end of a rigid radius under the following conditions: Length of radius, 6 feet; number of revolutions per second, 10?
Ans. 3,682.7 lb.

2. What is the pressure per square foot due to a stream of water flowing through a pipe that is rotating on one of its ends, the length of the pipe being 5 feet and making 2 revolutions per second?
Ans. 4,395.9 lb. pressure per sq. ft.

3. A ventilating fan is 25 feet in diameter, and the port of entry is 10 feet in diameter. What is the radial length of the blades?
Ans. 7.5 ft.

4. A fan is 25 feet in diameter, and the diameter of the port of entry is 10 feet. What is the length of the radius of gyration?
Ans. 9.5 ft.

5. A ventilating fan is 25 feet in diameter, the port of entry is 10 feet in diameter, and the radius of gyration is 9.5 feet. What is the velocity in feet per second of the center of gyration when the fan makes 45 revolutions per minute? Ans. 44.7678 ft. per sec.

6. The velocity of the center of gyration of a ventilating fan is 44.7678 feet per second, and the radial length of the blades is 7.5 feet. What is the total pressure generated by the fan to overcome the mine resistance, and to set the air in motion into and out of itself?
Ans. 11.3032 lb. per sq. ft.

TYPES OF FANS.

1043. Centrifugal fans are used for blowing and exhausting. Exhausting fans are in most general use, though there are many advocates and users of the blowing-fan. So far as mechanical efficiency is concerned, exhaust-fans and blowing-fans are practically equal.

The general principles of each are the same, except in reverse order. The exhaust-fan draws from the mine and discharges into the outer atmosphere, while the blowing-fan draws from the outer atmosphere and discharges into the mine.

1044. Prominent Types of Fans.—The most prominent types of centrifugal ventilators now in use in mining countries are four in number, the principal representatives of which are (*a*) *Waddle*, (*b*) *Schiele*, (*c*) *Guibal*, (*d*) *Capell*. These four will be described as representing the main features of all other forms, which are modifications of these original types.

1045. The Waddle Fan.—The characteristic features of the *Waddle* fan, Fig. 165, are the *curvatures of its blades*—backwards from the direction of their motion; their tapered form, tapering towards the circumference, and the tight box sides, which revolve with the blades. The blades leave the orifice of intake radially, but curve backwards from the direction of their motion till they are almost tangential at the circumference. The blades are so tapered from the orifice of intake to the circumference that the breadths of the blades at

FIG. 165.

to be equal to 50 square feet, and the area of the top of the evase chimney to be equal to 200 square feet. Now, the velocity of discharge at the top of the evase chimney will only be one-fourth of the velocity of the air blowing through the orifice of discharge of the fan, and as the resistances that arise when air at a high velocity strikes still air are proportionate to the squares of the velocities, the resistance due to the air leaving the top of the evase chimney is to the resistance of the higher velocity as 50^2 is to 200^2 or as 1 is to 16.

EXAMPLES FOR PRACTICE.

1. What is the centrifugal force in pounds due to a 5-pound weight revolving on the end of a rigid radius under the following conditions: Length of radius, 6 feet; number of revolutions per second, 10?
Ans. 3,682.7 lb.

2. What is the pressure per square foot due to a stream of water flowing through a pipe that is rotating on one of its ends, the length of the pipe being 5 feet and making 2 revolutions per second?
Ans. 4,395.9 lb. pressure per sq. ft.

3. A ventilating fan is 25 feet in diameter, and the port of entry is 10 feet in diameter. What is the radial length of the blades?
Ans. 7.5 ft.

4. A fan is 25 feet in diameter, and the diameter of the port of entry is 10 feet. What is the length of the radius of gyration?
Ans. 9.5 ft.

5. A ventilating fan is 25 feet in diameter, the port of entry is 10 feet in diameter, and the radius of gyration is 9.5 feet. What is the velocity in feet per second of the center of gyration when the fan makes 45 revolutions per minute? Ans. 44.7678 ft. per sec.

6. The velocity of the center of gyration of a ventilating fan is 44.7678 feet per second, and the radial length of the blades is 7.5 feet. What is the total pressure generated by the fan to overcome the mine resistance, and to set the air in motion into and out of itself?
Ans. 11.3032 lb. per sq. ft.

TYPES OF FANS.

1043. Centrifugal fans are used for blowing and exhausting. Exhausting fans are in most general use, though there are many advocates and users of the blowing-fan. So far as mechanical efficiency is concerned, exhaust-fans and blowing-fans are practically equal.

about the fan, to accommodate the flow from each compartment. The velocity of the air is thus made uniform all around the circumference, and each compartment furnishes its proportion of air in a continuous flow. In exhausting, the discharge from the casing is conducted to the expanding chimney, where its velocity is much reduced, before it is finally thrown out upon the atmosphere. In force or blower fans, the expanding casing should connect with the mine passages by a uniformly expanding fan drift.

1047. The Guibal Fan.—This fan revolves within a closed case and delivers its air into an involute chamber, which gradually expands into the evase chimney. The fan is illustrated by Fig. 167.

FIG. 167.

1048. The Capell Fan.—This fan is constructed somewhat after the type of the Guibal fan, with considerable additions and improvements, such as is illustrated by Fig. 168. All the centrifugal fans in use, however, may be put into two groups; namely, closed and open fans. Among those just noticed, the Waddle is an open fan; the Schiele, the Guibal, and the Capell are closed fans.

FIG. 168.

different distances from the center of the fan-wheel vary inversely as the radial lengths of the points at which the breadths are measured.

1046. The **Schiele fan,** Fig. 166, very much resembles the Guibal fan in its mode of action, although its construction is in some important respects quite different. In the Schiele fan a disk takes the place of the spider wheel in the Guibal fan, and this makes necessary the duplication of the blades, for they are attached on opposite sides of the disk. The disk makes a complete partition within the fan, and, therefore, the supply of air to the blades must come from two ports of entry, one for each set of blades. The Schiele fan

Fig. 166.

is a fast-running one, and, therefore, to do the same amount of work its diameter is only about one-third that of a Guibal fan of the same capacity. By reference to the figure, it will be seen that this fan is set within a spiral casing surrounding the circumference and leading to an expanding or evase chimney. The student must notice carefully the effect of the spiral casing surrounding the circumference of the Schiele fan, as it is a most important factor in fan construction. It provides a uniformly increasing sectional area

about the fan, to accommodate the flow from each compartment. The velocity of the air is thus made uniform all around the circumference, and each compartment furnishes its proportion of air in a continuous flow. In exhausting, the discharge from the casing is conducted to the expanding chimney, where its velocity is much reduced, before it is finally thrown out upon the atmosphere. In force or blower fans, the expanding casing should connect with the mine passages by a uniformly expanding fan drift.

1047. The Guibal Fan.—This fan revolves within a closed case and delivers its air into an involute chamber, which gradually expands into the evase chimney. The fan is illustrated by Fig. 167.

Fig. 167.

1048. The Capell Fan.—This fan is constructed somewhat after the type of the Guibal fan, with considerable additions and improvements, such as is illustrated by Fig. 168. All the centrifugal fans in use, however, may be put into two groups; namely, closed and open fans. Among those just noticed, the Waddle is an open fan; the Schiele, the Guibal, and the Capell are closed fans.

Fig. 168.

PRACTICAL VENTILATION.

QUANTITY, VELOCITY, AND CONDUCTION OF AIR.

1049. Use of the Air-Current.—The necessity for the air-current in mines is threefold, viz: (1) to furnish fresh air to the men and animals in the mine; (2) to sufficiently dilute and render harmless the poisonous and explosive gases of the workings; (3) to remove these by sweeping them from their lodging places or the cavities in which they are lodged.

1050. Efficient Ventilation.—The efficient or thorough ventilation of a mine is dependent upon three essential elements:

(*a*) Volume of the current.
(*b*) Velocity of the current.
(*c*) Manner of conducting the current.

1051. Volume of Air Required.—The quantity of air required to be furnished per minute to the workings of a mine is usually fixed by the law of the state or country in which the mine is located. The amount specified is usually 100 cubic feet per minute per man, and 500 cubic feet per minute per mule, in non-gaseous mines. In gaseous mines this amount is increased to 150 cubic feet per man, and, in some cases, as in the anthracite mines of Pennsylvania, 200 cubic feet per minute are required by law. This method of fixing the amount of air required is purely arbitrary, and often not adapted to the existing conditions, as the workings in a thin seam will have an abundance of air and, perhaps, too high a velocity, while in a thicker seam the velocity will be too low for proper ventilation.

1052. Velocity of the Current.—This is a very important factor in the ventilation of a mine, as upon it largely depends the removal of the mine gases. A body of firedamp or of marsh-gas which is exuding or has collected in some cavity of the roof, or at the face of some rise heading, will require a current having a certain velocity to dilute

it and drive it from its position. In like manner, the heavier gases, as carbonic acid gas, settling towards the dip workings and low places of the pit, can not be carried out if the velocity of the current is too low. The current velocity should not be permitted to fall below 3 or 4 feet per second at any working face.

1053. On the other hand, too high a velocity of the ventilating current is always objectionable, and, in gaseous mines, dangerous. The Anthracite Mine Law of Pennsylvania provides that all air-passages shall be of a sufficient area to allow the passing of 200 cubic feet of air per man per minute at a velocity not exceeding 450 feet per minute; and this velocity may be taken as a safe maximum limit in gaseous mines. It must be remembered that a safety-lamp carried against a current is virtually subjected to a velocity equal to that of the current plus the velocity with which the lamp is carried against the current. In non-gaseous mines, the velocity of the main intake may be anywhere from 10 to 20 feet per second without causing serious annoyance. In many cases it exceeds this amount. It should not fall below 3 or 4 feet per second in any airway in the mine.

1054. Conducting the Current.—The thorough ventilation of the working face in any mine is dependent to a large extent upon the manner in which the doors, stoppings, brattices, and overcasts, or bridges, are erected. These must not leak. **Doors** must be hung with a fall sufficient to close behind a passing car, and must be fitted tightly to a substantial frame. Where the ventilating pressure is light, a door is sometimes hung to swing both ways, to save trapping. Ordinarily, however, this can not be done, and the door must then be hung to open against the current. Canvas flaps are sometimes nailed to the bottom of a door to prevent leakage and allow of good clearance. Double doors are often used in cross-cuts near the shaft bottom between the main airway and the return. The object of double doors is to prevent the momentary stoppage of the ventilating current when a car is passing through the

cross-cut. The doors are set from 6 to 8 yards apart, or farther, if the length of the cross-cut will permit. **Stoppings** are commonly built of a double wall of slate, with a few inches of space between. This space is filled completely to the roof with fine sand or clay from the surface, or with dirt taken from the roads. Stoppings upon the main air-courses of large mines are often plastered with clay, or laid up with brick instead of slate walls. **Brattices** are partitions, usually of wood or canvas (brattice-cloth), for the purpose of dividing the airway for a short distance near the face into an intake and a return. A **curtain** is a heavy canvas hung across an airway or the mouth of a room to partially turn the air. A curtain serves as a regulator, inasmuch as it permits a portion of the air-current to pass it, while the remaining portion is forced into another passageway.

1055. Regulators are contrivances for regulating the division of the air between two or more airways. The usual form of regulator is shown in Fig. 147, and consists of a wooden brattice built across the airway and having an opening provided with a shutter, by which the size of the opening may be increased or decreased, and thus any desired division of the air secured. **Bridges** are built in airways for the purpose of crossing the air-currents. A bridge may be an **overcast**, as shown in Fig. 169, in which C is the coal-seam, G is the main airway, and R is an overcast cut out of the roof for the purpose of conducting a current of air over or under another current. If the cross-current is conducted under the main airway, it is called an **undercast.** In this case the *bridge* forms the floor of the main airway and the roof of the cross-cut. The dis-

FIG. 169.

advantages of the *undercast* are: (1) water is apt to close the passage; (2) the bridge is more difficult to keep air-tight, being subject to travel of mules and cars; (3) if the cross-current is the intake, the air is made unwholesome by the dust of travel being sifted down through the bridge floor. The bridge floor is usually made by laying a double thickness of plank upon cross-stringers of railroad iron or oak, and covering the whole with dirt from the roadways.

1056. Fig. 170 is an illustration of the practical arrangement of stoppings, brattices, and curtains. *A B* are a

FIG. 170.

pair of entries which struck a roll. The entry *A* was abandoned for the time, and *B* was driven ahead as a fore-winning or prospect entry. For this purpose a temporary brattice was erected from the outside rib of the last cross-cut at *A*, towards the face of the entry *B*, by setting a row

of props two feet from the rib of the entry, and nailing brattice-cloth or light boards to them. The air-current is then forced to travel towards the face before it can return to the cross-cut behind the brattice. An entry stopping and room stoppings are shown at *s*, *s*, *s*. Curtains are hung at the mouths of all the rooms, except the outside and inside ones, which are left open. A curtain or a door is then hung upon the entry at *D*, just inside the mouth of the first room, which deflects the current into the face of the rooms. A curtain at *D* will usually accomplish this, but if the room workings are extensive, a door should be used and an opening left in it, or at one side of it, sufficient for the ventilation of the portion of entry thus cut off from the current. This example will serve to illustrate a practical method of conducting a current of air through a mine. It is sometimes necessary to deflect the current so that it will sweep a particular cavity of the roof or point of the entry, where a dangerous body of gas would otherwise collect. Wherever this is necessary, attention must be given to it, as no system of ventilation will be efficient unless the air is made to brush the gas from all of its lodging places. The *volume* of the current may be sufficient, and it may travel at the required *velocity*, yet the ventilation of the working places will be poor, unless the air is properly conducted and sweeps the entire face and roof.

INSTRUMENTS.

INSTRUMENTS FOR MEASURING THE RESISTANCE OF AIRWAYS.

1057. The instruments for measuring the resistance of air are:

(*a*) *Pressure.*—Water-gauge and manometer.

(*b*) *Velocity.*—Anemometer.

The student has learned that the intake pressure of an airway is always greater than the return pressure. The difference of pressure is measured usually by means of a

water-column in one arm of a bent glass tube. This instrument is called a **water-gauge.**

1058. The Water-Gauge.—Fig. 171 shows the usual form of water-gauge in use in the mines. The scale is divided into inches and decimals of an inch and is movable, in order that its zero can be adjusted to the lower water-level by means of the thumb-screw below. The tube is bent into the form of a letter U, both arms being open at the top to the free admission of air. The left-hand arm at the top, however, is cemented into a brass tube which makes a square bend, passing through the wooden base to which it is secured, so that the open tube can be connected with the opposite air-current to measure the difference in pressure.

FIG. 171.

Fig. 172 shows the water-gauge in position upon a mine door or brattice, in a cross-cut between the intake and return airways. *C* is the intake and *D* the return of the mine. One of the open ends of the gauge-tube *A* is thus subject to the intake pressure, while the other end is acted upon by the return pressure.

FIG. 172

These pressures being unequal, their difference will be equal to the weight of water which is unbalanced in the gauge.

When the pressure of the air upon each end of the gauge is equal, the water will stand at the same height in each arm; if, now, the pressure be increased upon one end, the level of the water in that arm will sink, while it rises in the other an equal amount. Suppose that the intake pressure over that of the return is sufficient to cause the water-level to sink 1 inch in the arm open to the intake. The level of the water in the other arm will rise 1 inch. Now, by moving the scale until its zero corresponds exactly with the lower water-level, the reading of the upper level will be 2 inches. This will represent 2 inches of water-column, balanced only by the ventilating pressure of the mine.

1059. To calculate the pressure per square foot of area which supports this water-column: The weight of 1 cubic foot of water (62.5 pounds) corresponds to a pressure of 62.5 pounds per square foot for 12 inches of water-column; and 1 inch of water-column or water-gauge will, therefore, be equivalent to $\frac{62.5}{12} = 5.2$ pounds pressure per square foot. Hence, to calculate the pressure p in pounds per square foot of sectional area, when the water-gauge W is given in inches, the formula $p = 5.2\ W$ is used.

That is, *the unit of ventilating pressure or the pressure upon each square foot of the sectional area of an airway, in pounds, is 5.2 times the reading of the water-gauge in inches.*

1060. The reading of the water-gauge must always be taken between the intake and the return current, and as near the mouth of the return current as possible, in order that it shall express the full resistance of the mine.

1061. The Anemometer.—This is a wind-gauge for measuring the velocity of a current in an airway by timing the revolutions of the vanes. Fig. 173 shows the most reliable form of anemometer. A is a wind-wheel whose revolutions are indicated by the registering dials at B. These dial-hands are so geared to one another and to the spindle, or axle, of the vane, that the divisions upon each dial represent 10 revolutions of the next preceding dial-hand, and each

division of the large circle corresponds to 1 revolution of the vane. There are 100 divisions in the large circle; and, therefore, 1 revolution of the large hand denotes 100 revolutions of the vane. One revolution of the dial-hand *c* denotes

FIG. 173.

10 revolutions of the large hand *B*, or 1,000 revolutions of the vane. Thus, the dials in Fig. 173 register 2,118 revolutions of the vane.

1062. The vanes are inclined at such an angle that 1 revolution of the vane corresponds to 1 foot of travel of the air in the airway. There is a disconnecting device shown near the handle by which the registering dials can be instantly thrown out of gear, which makes it possible for the operator to take more accurate readings One important point to be borne in mind, in taking careful measurements of the current passing in an airway, is that the velocity is

not the same in all parts of the passage. The friction of the current upon the sides and top and bottom of the airway retards the air nearest to these surfaces. As a consequence, the air rolls, as it were, in whirling circles upon these surfaces. The velocity of the current is, therefore, greatest at the center of the passageway and least in the corners.

1063. In Fig. 174 is illustrated a common method of obtaining a fair average reading for the entire area of the entry. The passageway is divided into 9 equal squares by the imaginary vertical and horizontal lines *a a*, *a a*, etc. The anemometer is held in each of the outer squares for the same length of time, say 15 seconds, moving it from one to another in regular succession, and is then held in the center square for a period as long as that of the other 8 squares combined, thereby occupying $\frac{8 \times 15}{60} \times 120 = 240$ seconds, or 4 minutes. Suppose, when this has been carefully done, the reading of the anemometer is as shown in Fig. 173 (2,118); then the average velocity for the entire area of the airway would be $\frac{2{,}118}{4} = 529\frac{1}{2}$ feet per minute. The quantity of air q passing in the airway per minute is calculated by the formula

FIG. 174.

$$q = a v.$$

When using the anemometer, the operator should endeavor, as far as possible, not to obstruct the passageway or contract its area by his body. He should stand to one side of

the center of the passage and make allowance for his body, especially when the sectional area of the airway is small. The anemometer should be held at right angles to the direction of the current. Consideration must also be given to the fact that an air-current, like a water-current, moves in channels. The velocity of the air will always be found very much greater along the rib of an airway which corresponds to the outer circle of a bend. The instrument held by this rib will often show a good velocity, while close to the other rib there is scarcely sufficient motion to obtain a reading. In fact, velocity measurements should be taken, if possible, where the passage is straight and smooth.

INSTRUMENTS FOR MEASURING DENSITY OF AIR.

1064. The instruments for determining the density of air are :

(*a*) The *barometer*, for measuring atmospheric pressure.

(*b*) The *thermometer*, for finding the temperature of the air.

The density of air depends mainly upon two factors, barometric pressure and temperature. When these are known, the weight of 1 cubic foot of air is easily determined by formula **57.**

1065. The Barometer.—This instrument has been previously described in another paper, and will only be referred to here. Its use is important in all mining operations, where mines are opened on an extensive scale. It is often found in the offices of large mining companies, connected with a continuous self-registering apparatus that records the barometric height for every hour.

1066. The Thermometer.—The temperature of the air is determined by the thermometer, which is a glass tube having a small bulb blown upon the lower end. The bulb and a portion of the tube are filled with mercury, the upper end being closed, after boiling the mercury, to expel the air. The tube is attached firmly to a base, as shown in Fig. 175, which is then graduated so as to mark the degrees of

temperature by the expansion of the mercury, which rises and falls in the stem.

There are two thermometer scales in common use, the Fahrenheit scale, marked F in the figure, and the Centigrade scale, marked C. They differ principally in the location of the zero mark.

The **Centigrade scale** is a decimal scale. Its zero is marked by the freezing-point of water, while the boiling-point is marked 100°.

The **Fahrenheit scale** is the scale most used in America and in England. The freezing-point of water is marked 32° above zero, and the boiling-point 212° above zero.

1067. By comparing these two scales, it can be seen that 100° on the Centigrade scale correspond to $212 - 32 = 180°$ on the Fahrenheit, or 5 degrees C. = 9 degrees F. Hence, to convert any Centigrade reading into the corresponding Fahrenheit reading, use the following formula:

$$F = \tfrac{9}{5} C + 32, \qquad \textbf{(76.)}$$

in which F is the Fahrenheit reading and C the Centigrade reading. That is, $\frac{9}{5}$ *of any Centigrade reading, plus 32, is equal to the corresponding Fahrenheit reading, attention being paid to plus and minus readings, when above or below zero, respectively.*

Fig. 178.

To convert any Fahrenheit reading into the corresponding Centigrade reading, use the following formula:

$$C = \tfrac{5}{9} (F - 32), \qquad \textbf{(77.)}$$

in which the letters have the same meaning as in formula **76.** That is, *from the given Fahrenheit reading subtract 32, and take $\frac{5}{9}$ of the remainder; the result will be the corresponding Centigrade reading.*

NOTE.—In each of the two preceding rules, all readings, of either scale, above zero are plus, and all below zero are minus. A few examples will make the method clear.

Example.—Convert 50° C. into the corresponding Fahrenheit reading.

Solution.—Using formula **76**,

$$F = \tfrac{9}{5} \times 50 + 32 = 122° \text{ F. Ans.}$$

Example.—Convert $-10°$ C. into the corresponding Fahrenheit reading.

Solution.—Using formula **76**,

$$F = (\tfrac{9}{5} \times -10) + 32 = -18 + 32 = 14° \text{ F. Ans.}$$

Example.—Convert $-30°$ C. into the corresponding Fahrenheit reading.

Solution.—Using formula **76**,

$$F = (\tfrac{9}{5} \times -30) + 32 = -54 + 32 = -22° \text{ F. Ans.}$$

Example.—Convert $-4°$ F. into the corresponding Centigrade reading.

Solution.—Using formula **77**,

$$C = \tfrac{5}{9}(-4 - 32) = \tfrac{5}{9} \times -36 = -20° \text{ C. Ans.}$$

The student will notice that in these formulas, 32 is added and subtracted algebraically; that is, when the signs are like, the quantities are added together, their sum having the same sign; but when the signs are unlike, the lesser quantity is subtracted from the greater, and the remainder takes the sign of the greater. Thus, in the solution of example 2, where $+32$ is added to -18, subtract 18 from 32, and the remainder, 14, takes the plus sign. In example 3, subtract 32 from 54, and the remainder, 22, takes the minus sign.

EXAMPLES FOR PRACTICE.

1. What temperature Fahrenheit corresponds to 100° C.?
Ans. 212° F.

2. Convert 290° Centigrade into the corresponding Fahrenheit reading.
Ans. 554° F.

3. What reading upon the Centigrade scale corresponds to 5 Fahrenheit?
Ans. $-15°$ C.

4. Convert $-40°$ Fahrenheit into the corresponding Centigrade reading.
Ans. $-40°$ C.

AIR COLUMNS.

1068. In speaking of the *natural* means at hand for producing ventilation, *heated air columns* have been treated. It is important to notice that since air has weight, *all* columns of air exert a downward pressure upon the area of

their base equal to the weight of the column. Also, the downward pressure of air columns creates an *equal* pressure in every direction upon the air of each level of the airway. That is, the pressure throughout each level section is the same; but if the elevations of the sections are different, the pressure in each section will be different. The pressure per square foot of sectional area in the airway due to any air column is equal to the weight of a column of air of the same height and having a uniform section of 1 square foot from bottom to top.

1069. Air columns may be vertical, as in the case of the furnace shaft $B\,A$, Fig. 176, or inclined, as in the case of the slope air column $C\,E$. In either case, the pressure at the base upon 1 square foot of sectional area, due to the weight of the air column, is calculated from its *vertical height*. In the

figure, the depth of the shaft D is equal to the vertical height of the slope, and for the same temperature the pressures per square foot of sectional area, due to these two air columns, are equal.

1070. As explained previously, the difference in the weight of two air columns connected by an airway at their bases causes the air to flow through the airway, from the heavier towards the lighter column. It can readily be seen that the weight of one of these columns is **positive**, while that of the other is **negative**, so far as concerns the motion of the air-current. The weight of the *positive* column always acts in the direction in which the current moves in proportion to its excess of weight over the negative column. Thus, the excess of weight of the positive air column causes the flow of the current.

1071. It is a matter of common observation in mines that **rise workings** are more difficult to ventilate than **dip workings**, when the bottoms of the shafts are on the dip

side of the workings. The reason for this is found in the fact that the intake air, or that flowing to the rise, is always cooler and denser than the return air, which has become heated by the higher temperature of the workings, and which must flow down grade to the upcast. The effect of this will be to *retard* the ventilation of the workings.

On the other hand, if the intake runs to the *dip*, as in dip workings, and the continuous flow of the air is upward to the upcast, there will result a *heavy* positive column and a light *negative* column; the combined effect of these will *assist* the ventilation.

The influence of dips and rises in mine workings is thus seen to be a powerful one, in fact often completely controlling the ventilation of the workings. For this reason, seams having any considerable inclination should be so ventilated that as far as practicable the course of the current will be towards the rise. This is known as **ascensional ventilation,** and is an important consideration in the ventilation of all mines.

1072. As previously explained, the weight of the **motive column** is the excess of weight producing a flow of air; hence, it is the algebraic sum of the weights of all the air columns, positive and negative. It is often convenient to reduce the various factors in any ventilation to a single *motive column*, which at once expresses the height of air column whose weight produces the ventilating pressure.

1073. An essential point, in regard to all air columns, is the *density* of the air which forms it. Whatever affects the density of the air affects the weight of the column. Temperature, pressure, moisture, presence of gases, all affect the weight of the air column, to a greater or less degree. In nice calculations, it would be right to consider all these factors for each column respectively. In ordinary calculations, however, it is customary to consider the temperature of the column and the barometric pressure only, ignoring the mine pressure, the amount of moisture in the air, and the presence of gases. The latter, excepting

moisture, are often very important factors. Mine pressure may affect the motive column to the amount of 2½ per cent., always increasing the density of the intake air. The presence of carbonic acid gas in the upcast current, or in the return current of any ascensional ventilation, acts to reduce the motive column, and may amount to as much as 20 per cent.

THE BEST METHODS OF VENTILATING GASEOUS AND NON-GASEOUS MINES.

1074. Ventilation of Flat, Non-Gaseous Seams.— Fig. 177 shows a plan of underground workings in a non-gaseous mine, worked upon the pillar and chamber method. The seam lies flat, or nearly so. The main feature of the ventilation here shown is the manner of splitting the air at each pair of cross-entries, so as to give to each pair a separate current. Or, one split of the air may be made to ventilate two pairs of entries near the face of the workings. On account of the expense, overcasts are never put in at any pair of cross-entries until the development warrants the outlay. It is evident, from observing the plan, that each overcast saves either a door or a stopping, and always leaves the main road free of doors. It must be remembered, however, that the practical limit to splitting an air-current is the velocity of the divided current, which must not fall below 3 or 4 feet per second in non-gaseous mines, and 5 or 6 feet per second where gas is given off. For example, if the sectional areas of the airways are each, say, 50 square feet, and there is only 15,000 cubic feet of air passing upon the main airway, this current can not be split, as its velocity now is but 5 feet per second, and another split would reduce it below the limit.

1075. Another important feature, in the practical ventilation of a mine, is the location of the stables. They must be situated close to the bottom of the shaft, so that the mules can be easily rescued in case of accident, and where the daily feed and refuse can be readily handled. It

is also essential that they be ventilated by a separate split of fresh air, as shown in Fig. 177, and that the return

Fig. 177.

current from the stables should pass through a regulator, to prevent excessive drafts, and thence directly out to the

upcast shaft, and not contaminate the air of the mine by mixing with it. In Fig. 177 the feed and refuse are handled through the door shown in the first cross-cut, next to the shaft. The mules enter the stable at the other end, which requires no door.

1076. Another important feature to be considered, in the ventilation of all mines, is the arrangement of the haulage roads with respect to the air-currents. It will be observed in Fig. 177 that the main haulage roads are made the *return* airways of the mine. This is the better plan in all non-gaseous mines, for two principal reasons: (*a*) freedom from dust upon the *intake;* (*b*) freedom from ice in the hoisting shaft in winter.

1077. Ventilation of Gaseous Seams.—In gaseous mines, the haulage is always of necessity done in the intake airways, to lessen the liability of explosion. If the mine represented in Fig. 177 were gaseous, it would be necessary to reverse the current, and cause it to circulate in a direction opposite to that shown, making the *hoisting* shaft the *downcast*. This is usually done by means of an exhaust-fan placed at the mouth of the *upcast* shaft. The doors throughout the mine would all require to swing in the opposite direction, but in other respects the arrangements in the two cases are identical, except only in respect to the velocity of the current, as already explained.

In very fiery mines, the main roads, or gangways, are often driven triple instead of double. This method is called the "Triple Entry System," a section of which is shown in Fig. 178.

The middle entry is always made the intake and haulage road, the two outer entries being the return for each side of the mine, respectively.. The *exhaustive* system of ventilation must be used in this case as in every other case where the *haulage road* is made the *intake;* otherwise, a door would have to be placed upon the haulage road, and there should then be two doors to prevent the stoppage of the current while the trip is passing. Such doors have been

introduced into the hoisting shafts in certain cases. They were made to work automatically and reciprocally, the one opening after the other had closed, each time the cage passed. This arrangement is complicated, and should never be used where it can be avoided.

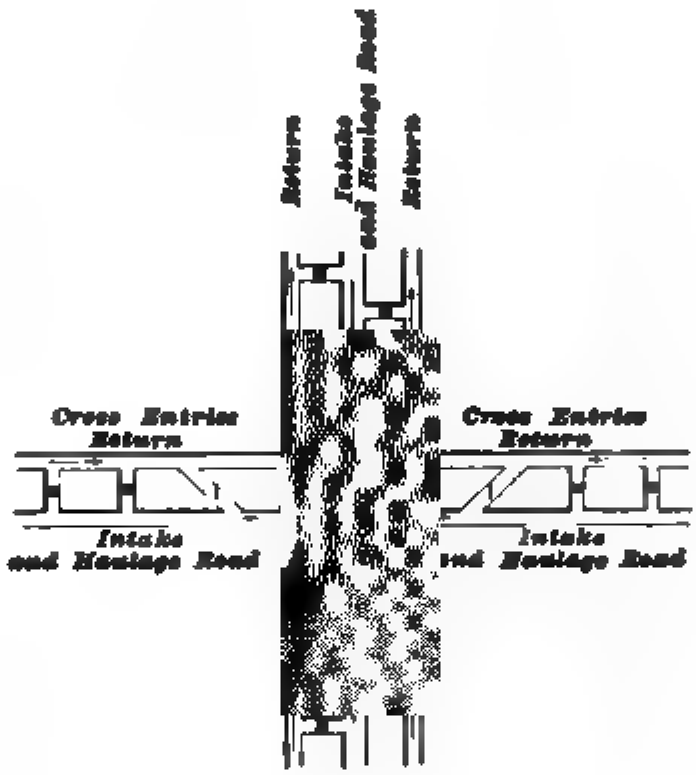


FIG. 178.

The chief advantage of the triple-entry system is that it furnishes a separate return for each side of the mine, and in case of an explosion, there is more complete isolation of the affected district.

1078. Ventilation of Inclined Seams.—The main point to be considered, in the ventilation of all inclined seams, is that it shall be *ascensional.* In other words, the *intake* air being, as a rule, cooler than that of the return, should be conducted at once to the lowest portion of the workings, whence, as it gradually absorbs heat from the mines, it tends to rise. The natural heat of the mine, as has been previously explained, always creates a small *motive column*, which *assists* the ventilation when the cooler intake runs to the dip, and its return, to the rise. In general, the cool outside air *falls naturally* to the lowest place in the mine, and as it becomes heated in its passage through the workings, it *rises naturally* to the higher portions. As far as it is practicable to do so, this principle of ascensional ventilation must be applied in conducting an air-current through workings in an inclined seam.

1079. Fig. 179 shows the ventilation of the workings of an inclined seam opened by a slope. The *intake* is a

shallow shaft near the mouth of the slope, while the return is the main haulage road leading to the mouth of the slope. The first four pairs of entries, it will be observed, are ventilated by a separate split. Each of these splits is conducted through the lower entry of the pair, directly to the face of the entries, and after passing through the inside cross-cut, it enters the last room, being deflected, if necessary, by a canvas hung on the entry. The current traverses the working face of each room by passing through the cross-cuts in

FIG. 179.

the room-pillars, and along the edge of the gob where the pillars are being drawn back.

If the seam represented in Fig. 179 were a gaseous seam, the direction of the current should be reversed, the main haulage roads being made the intake airways for each respective section of the mine. This is best done by making the fan at the air-shaft exhaust.

The types of ventilation, or the manner of conducting the air through a mine, illustrated in these general plans, cover

all cases of practical ventilation. The successful application of the principles involved in any one particular case will, however, often depend, owing to varying conditions, upon the skill and ingenuity of the man in charge.

1080. Manner of Handling a Body of Gas.—The removal of a body of gas that has accumulated in an idle room or unused chamber requires great precaution if the gas is fiery. This should never be attempted until all the men who are in the neighborhood of the outgoing air-current have been withdrawn. Brattices of canvas should then be so hung as to direct into the workings where the gas is lodged the main portion of the air traveling along the airway. Or, if the gas has accumulated in some cavity of the roof, the current must be made to sweep this cavity by a brattice erected in the entry beneath it. Great care is necessary not to ignite the gas. Only the most reliable safety-lamps should be employed, and they should be placed in fresh air, at a safe distance away, and carefully watched.

The practice, already alluded to, of passing a gaseous current over a furnace by first diluting it with a sufficient quantity of fresh air direct from the downcast, is a dangerous one. Indeed, best mining practice to-day does not tolerate a furnace in a mine that yields gas.

1081. Entrance of a Mine After an Explosion.—This part of the subject will be considered only as it depends upon the restoration of the ventilating current in the airways and workings. The call for volunteers and their organization into two or three rescuing parties, each under its own efficient leader, is followed immediately by the adoption of such measures as a hasty examination shows to be necessary to restore ventilation.

If an exhaust-fan was in use at the mine opening previous to the explosion, it will generally be found to be less injured by the force of the explosion than would be the case with a blower-fan, depending, of course, upon the location of the initial explosions with respect to the upcast and downcast shafts. The force of an explosion is usually

exerted in the direction of the intake opening. Any injury to the fan that incapacitates it for use must be speedily remedied. The original course of the ventilating current should not be altered, except upon the most urgent demands.

As quickly as the intake current begins to enter the mine, the men should follow, equipped with good lamps, picks, shovels, saws, axes, sledges, brattice-cloth, and boards. They proceed at once to follow the air, rebuilding doors and stoppings or erecting a temporary line of brattice for conducting the air around a fall. The main point to be borne in mind is that no effectual advance can be made ahead of the air.

1082. Mine Fires.—This term applies to any form of slow or rapid combustion taking place in the passages or workings of a mine. Mine fires are a dangerous element in *any* mine. In gaseous mines in particular is this the case, the presence of a fire being an imminent source of peril.

The chief **causes** leading to mine fires are: (*a*) *spontaneous combustion*, as it is called, or combustion from natural causes, arising from the storing of slack and fine coal in the gob; (*b*) *ignition of the coal* by a gas-feeder fired by the flame of a blast; (*c*) *ignition of a door-frame, brattice, or timbers* by a naked lamp. Whatever the cause, a fire in the workings or passages of a mine should receive prompt attention. Its presence is manifested not more by the heat developed than by the peculiar odor imparted to the air.

1083. Treatment of Mine Fires.—The treatment of mine fires will be considered with particular reference to the ventilating current. The manner of treatment is divided into four classes, according to the stage of development the fire has reached, viz.:

(*a*) Direct method, with hose and water or portable chemical fire-extinguishers.

(*b*) Loading out in mine cars.

(*c*) Isolating from the air by special stoppings, so built as to effectually prevent air leakage.

(*d*) Flooding.

1084. In its incipient stages, a mine fire can be extinguished by the direct method, and a gob fire especially can readily be loaded out in mine cars; but it frequently happens that the fire assumes larger proportions before it is detected.

The first method of treatment is a simple one, and needs no explanation except to state that efficient chemical fire-extinguishers are on the market, and that the simple methods of operating them are fully explained by the manufacturers.

1085. When it becomes necessary to build stoppings for the isolation of a mine fire, much care is needed, both in the location of the stoppings and in the order in which they are erected. The places chosen as the locations of stoppings should be, as far as practicable, in the narrowest openings available in the solid coal. The affected area should be completely shut off and sealed to the access of air. The stoppings must be well built, and of the quickest available material. Care must always be taken to begin sealing off a fire at its side next the return air and work towards the intake, sealing the intake opening last. The reason for this is that, by closing the return-air side first, the gases set free by the fire drive back the fresh air after that stopping is made, and when the intake is properly closed there is no chance for imprisoned pure air to initiate an explosion. It also affords a better opportunity for the dilution of the gases with the air of the ventilating current. Care, however, must be taken to avoid breathing the carbonic oxide, or white damp, incident to mine fires, as it is the most poisonous gas known.

In a gaseous mine it would be unsafe to proceed in any other way than that above described. Violent explosions have been known to result from neglect of these precautions. From the moment of the sealing of the inlet end, if this end be sealed first, the gases generated by the fire, and

otherwise, increase in volume and move slowly towards the outlet, where they have free access to the airway in a dangerous form.

When these stoppings have remained closed for a sufficient period, and it is necessary, in order to work the coal, that they be opened, it must be done with the utmost caution, and in the reverse order to that in which they were sealed. The stopping at the intake end is thus the first one to be taken down, as it was the last one put up. Careful search must at once be made to discover any smouldering remains of the fire, and for days after the place must be closely watched.

1086. Flooding a mine, in order to extinguish a fire, is only considered as a last resort. At times, certain portions of the mine which alone are affected are shut off and flooded. It then becomes necessary to construct dams sufficiently strong to withstand the pressure due to the water. The construction of these dams is treated particularly in Methods of Working.

HOISTING AND HOISTING APPLIANCES.

MOTORS.

INTRODUCTION.

2451. In order that we may study hoisting machinery to the best advantage, we will divide it into its component parts and consider these separately. They are as follows:

Motors,	**Cars,**
Drums,	**Rope Carriers,**
Ropes,	**Tracks.**

In other words, a hoisting plant, generally speaking, consists, first, of a *motor* to supply power and to rotate a drum; second, of a *drum* to be driven by the motor and to wind upon itself a rope; third, of a *rope* to be wound upon the drum and to carry at its end a car; fourth, of a *car* to be carried by the rope and to contain the material to be hoisted; fifth, of *rope carriers* to guide and control the rope; and, sixth, of a *track* to guide and control the car during its travel.

2452. As we have said, the motor of a hoisting plant is that part which supplies the power. In the case of an ordinary **windlass,** as shown in Fig. 888, the motor is a man, who turns the crank-handles, and thus applies power

Fig. 888.

to the drum. In the case of a horse-gin, as shown in Fig. 889, the motor is a horse, which is hitched to the end of the horizontal lever, and, by walking around the gin, supplies power to the drum. These motors answer their purposes, but they are only capable of exerting small amounts of power; as it becomes necessary to hoist heavy loads from great depths,

Fig. 889.

more powerful motors must be employed. In Fig. 890 is shown an electric motor, or dynamo, arranged as a motor for a hoist; in Fig. 891 is shown a duplex or double-cylinder

Fig. 890.

engine similarly arranged. We have before us, then, two powerful motors applicable to a hoisting plant; namely, electric motors and engines. In actual practice, these are found to fill all requirements, and, under certain conditions, each has advantages over the other.

It would be well to note here that there is a fundamental difference between the first two motors cited and the last two. The first two—that is, the man and the horse—actually supply the power that is transmitted to the drum. The last two—that is, the dynamo and the engine—do not actually supply the power, but simply transform it. The electric motor is supplied with electrical energy from some other source and transforms it into mechanical power, which is then transmitted to the drum. The engine is supplied with energy in the form of steam or compressed air, and

Fig. 801.

transforms it into mechanical power. As far as the hoisting plant is concerned, however, these motors, the electric motor and the engine, do supply the power.

ELECTRIC MOTORS, OR DYNAMOS.

2453. Of late years electricity has found its way into almost every branch of engineering, and mining engineering is not an exception to the rule. We find it taking the place of steam and compressed air in driving drills and mining machines, locomotives for underground haulage, pumps, and hoisting-engines. For the latter purpose it is well adapted and, under certain conditions, has many advantages. Some of these are due simply to the use of an electric motor in place of an engine; others are due to the use

of electricity instead of steam; and still others are due to the system of electrical transmission and transformation itself.

2454. By using an electric motor for a hoist a rotary motion is obtained directly, and can be reduced and transmitted to the winding drum simply through spur-gearing; whereas an engine gives a reciprocating motion that must be transmitted to the drum by means of cross-heads, connecting-rods, and cranks. The reason why the motion of an electric motor must be reduced, in transmitting it to the drum, is because the speed is necessarily too high for the drum. An electric motor is a less complicated piece of mechanism than an engine, and it consequently requires less attendance and less repairs. It has no valve-gear to get out of order, and it is more compact and occupies less room than an engine of the same power. The speed and power of an electric motor are regulated by means of a controller as readily as the speed and power of an engine are regulated by the throttle.

2455. By using electricity instead of steam the necessity of steam-pipes is avoided, their place being taken by two wires. These are much more easily laid and carried than steam-pipes. They take up little or no room, and can be carried along the most tortuous passageways when the power is wanted underground. There is no heat from them as there is from steam-pipes, and the loss due to the resistance of the wire to the current, known as *line loss*, is very small compared with the loss due to the condensation of steam in the pipes. If the hoist is to be underground, there is no exhaust-steam to heat and vitiate the air and rot the timbers.

If the power has to be transmitted any considerable distance, or if it must be transformed from some natural source, as wind or water power, the advantages of the electrical system of transmission and transformation are very great. Suppose, for instance, that we have a well-equipped mining plant consisting of boilers and engines, with power to spare, and it is desired to sink a shaft at some remote

place on the surface or at some underground point. We have simply to connect a dynamo to one of the engines to transform some of the spare power into electrical energy, run wires to where the hoisting must be done, and use an electric hoist. Or, suppose a water-power is available, either close at hand or at some distance from where the hoisting is to be done. We would then use a water-wheel or turbine and a dynamo instead of boilers and engines, and transmit and use the electrical energy as before. By such an arrangement, we would save the cost of fuel and the handling of it and the ashes.

2456. From the foregoing, it is evident that the student of hoisting machinery should know something of electrical matters, and be as familiar with the fundamental laws of electricity as he is with those of steam, and as familiar with the characteristic points of a dynamo as he is with those of an engine.

2457. There are two kinds of dynamos: viz., *direct-current* dynamos and *alternating-current* dynamos.

Alternating-current dynamos are of two kinds, *single-phase* and *multiphase*. In speaking of dynamos specifically, the one that transforms mechanical energy into electrical energy is called the generator, and the one that transforms the electrical energy back into mechanical energy is called the motor.

2458. Direct-current dynamos are suitable for hoisting machinery provided the hoist is near the generating station or plant; but if the current must be carried a long distance, the system becomes impracticable because of the necessarily low voltage of the current and the consequent great cost of the conducting wires. Direct-current dynamos, suitable for power purposes, can not be made to operate successfully at a much higher electromotive force than 1,000 volts, on account of the *arcing* and *short-circuiting* of the commutator and its connections. Furthermore, the direct current can not be transformed to a higher voltage except in a machine like a dynamo and having the same

objections. The plan of connecting up several dynamos in series, and so increasing the voltage, has been tried; but it is suitable only where power must be transmitted and used in large units. It would certainly be impracticable to connect up several generators in series to give a current of high enough voltage to be transmitted economically, and then use several motors in a similar series to drive a hoist.

2459. The single-phase alternating-current dynamo can without difficulty be made to generate and use a current of 3,000 to 4,000 volts, or even more if necessary, because the current is taken from and by two continuous rings without being rectified, thus avoiding the difficulties attending the commutators of the direct-current machine. By the principle of induction, an alternating current of moderate voltage can be transformed into a current of smaller amperage and higher voltage, for transmission, and can be re-transformed at the other end of the line to any voltage desired, the amperage varying inversely as the voltage. The energy remains the same, with the exception of a small loss in the transformation, amounting to about 2 per cent. As the coils of the transformer are stationary, and as there are no sliding contacts, any desired amount of insulation can be used, and almost any voltage that can be controlled on the line can be obtained. Many plants are in operation with currents of 10,000 to 12,000 volts, and some with currents as high as 50,000 volts.

The single-phase alternating current, however, consisting as it does of a simple alternating wave, is not suitable for hoisting machinery, because no satisfactory single-phase alternating motor has yet been devised that is self-starting under load and capable of speed regulation. If a motor built on the same lines as a single-phase generator is brought up to the proper speed by some external power, so that the alternating impulses will act in the right direction at the right instants, and if the current is then turned on and the load gradually applied, it will run satisfactorily at constant speed. Such a machine is called a synchronous motor, because it

runs synchronously, or in step, with the alternations of the current. Its speed can not be regulated, and if a sudden load causes it to slow down or lose step, it stops. It is inconvenient, and, in fact, impracticable, for service where frequent stops and starts are necessary, because starting it is such a tedious operation, and if it must start with the load on, it can not be used at all.

2460. The successful development of the multiphase system during the past few years has solved the problem, and has secured the advantages of both the direct and alternating currents. A multiphase generator has several windings, so placed as to generate several alternating currents differing in phase; that is, passing the zero and maximum points at different instants. Under the influence of these currents, which may be compared roughly to the cranks of a duplex or triplex engine without any dead-center, multiphase synchronous motors are self-starting under light load, while non-synchronous or induction motors will start under full load and are capable of speed regulation. The latter possess the good qualities of the direct-current motors, and the additional advantage of having no commutator. Furthermore, the multiphase alternating current, like the single-phase current, retains the indispensable quality, for long-distance transmission, of being transformable from low to high voltage for transmission, and from high to low voltage for use at its destination.

Most of the earlier dynamos used for hoisting were street-car motors geared to friction hoists. This type is very satisfactory for small or medium sized hoists, as the friction-gear is an assistance to the motor controller in smooth starting. For large hoists a positive-geared motor is more reliable; but it is desirable to interpose a friction-clutch or an equivalent device at some point between the armature and the drum as a safeguard against excessive strains on the gearing due to the inertia of the armature and the too sudden stopping of the drum with the brake.

2461. The choice of the best kind of motor depends to some extent on the size of the hoist, its location, and the

nature of the work. For an unbalanced hoist of moderate size, especially if placed underground and exposed to dirt and water, the iron-clad series-wound street-car type is well adapted, as it is strong, well protected, and designed to stand heavy work on intermittent service In this motor, efficiency, low heating, and absolute freedom from sparking are, to some extent, sacrificed for compactness and lightness. For large hoists, which are generally located in comparatively clean, dry places, which are either double-acting and balanced or single-acting and overbalanced, so as to act continuously, and in which high efficiency is of considerable importance, the stationary type of motor is usually preferable.

2462. The speed controller is one of the most important features of an electric hoist. On many of the earlier hoists, the commutated field, thrown into various combinations of different resistances by a cylinder switch, was employed, this form of control being at that time widely used in street-car service. This controller gave quite satisfactory results when assisted by friction-gearing, but with positive gearing it would not give a sufficiently gradual start. On most hoists, a variable resistance in armature circuit is employed; and by making this resistance sufficiently high, a perfectly smooth start may be obtained, even with a slack rope. The most satisfactory rheostatic controller, especially for heavy work, is one in which the resistance is cut in and out by a cylindrical switch with a magnetic blow-out, which avoids the troublesome effect of arcing at contacts, when the current is broken. In some cases it is practicable to use a double-motor equipment, with series-parallel controller, such as is now employed almost exclusively in street-car work.

ENGINES.

TYPES OF HOISTING-ENGINES.

2463. The engines of a hoisting plant are operated according to two methods. They are either continuous or intermittent in their action; that is, they either run continuously, empty or under some other load, and have the

work of hoisting put upon them by means of a clutch, or they are connected to the drum, either direct or through gearing, and are run only when making a hoist. The first of these methods has been employed extensively in regions where fuel is expensive, and where it is, therefore, desirable to obtain power economically by concentrating it. If a mine is equipped with an air-compressor for compressing air for rock-drills, or an underground hoist, or both, with a fan for ventilation, and has several openings where hoisting is to be done, one high-duty engine can be used to operate them all, and greater economy of fuel be obtained than if smaller and separate engines were used for each operation. The fan and the air-compressor would give a steady load, and the work of hoisting would be added to this as the hoists were made. But this kind of engine does not come naturally into our present studies, because it is an engine pure and simple, and is not affected by any of the requirements of the hoisting service. The clutches that are used for driving the drums will be taken up later in connection with drums.

The second method of using an engine in a hoisting plant —that is, of having it connected to the drum either direct or through gearing, and running it only when making a hoist—is the method generally adopted and the one we shall study, because it requires a hoisting-engine.

A hoisting-engine should be a *duplex engine*; it may be *condensing or non-condensing*; its size should bear an intimate relation to the amount and kind of work to be done; it may be driven by *steam* or compressed *air*; it should have a suitable, quick-acting *throttle-valve*, and the cylinders should be supplied with *relief-valves*.

2464. Duplex Engines.—One of the essentials of a good hoisting-engine is that it shall be capable of picking up the maximum load at any point in the hoist. If the engine is a single-cylinder engine, there will be two points in its revolution, the front and back dead-centers, at which it will be entirely powerless; and near to these points it will be capable of exerting only a very small amount of power.

If, with this sort of engine, the cage is at the bottom of the hoist, or, in fact, anywhere throughout the hoist where it might be necessary to stop, and the engine is on or near one of its dead-centers, it will be impossible to start to hoist. To avoid this state of affairs, two engines are placed side by side, acting on the same shaft, either engine being large enough to pick up the maximum load by itself. Such a combination of two single-cylinder engines is called a *duplex engine*. A cross-compound engine with its cranks set at right angles to each other will evidently produce the same effect in starting a load as a duplex engine; that is, there will be no dead-center. The two cylinders, each having its own piston and piston-rod, should have separate connecting-rods and cranks, and the cranks should take hold of the same shaft at right angles to each other. By this arrangement, it will readily be seen there is no dead-center or point where the engine is powerless. When one crank is on a center, the other is in its most effective position.

2465. Condensing or Non-Condensing Engines.— Steam-engines for hoisting service may be built either condensing or non-condensing; which is the better depends upon the circumstances of each case. Condensing engines are larger and cost more to build than non-condensing engines of the same power, and the only gain is in the greater economy of fuel of the condensing engines. If, therefore, the hoisting plant under consideration is to be located at a coal mine or in a coal region, where fuel is cheap, it would not be advisable, generally speaking, to go to the expense of putting in condensing engines. On the other hand, if the plant is to be located at a metal mine, where fuel is scarce, the economy of the condensing engine may more than balance its greater expense. It must be borne in mind, however, when deciding between the two, that hoisting is not an operation in which much economy can be gained by condensing, because the power required is so variable and intermittent. Condensing engines, too, are more complicated than non-condensing engines, and this is

always an objection. One of the essentials of a good hoisting-engine is that it shall be capable of exerting its maximum power at any time. In the case of a condensing engine, this demands two things. One is that provision must be made to admit live steam into the low-pressure cylinder, and the other is that the condenser and air-pump must be independent. We have assumed here that any condensing engine which we may consider must be a cross-compound; in the first place, because in a condensing engine the range of temperature of the steam is so great that with a single cylinder the initial condensation would be very large, and, in the second place, because, as we have seen, a hoisting-engine should be a double-cylinder engine in order that it may be able to start the load at any position of the hoist. This last reason was explained above, but the first reason will be considered more fully here.

2466. Suppose, for instance, an engine is supplied with steam at 100 pounds gauge pressure, which is equivalent to about 115 pounds absolute pressure. The temperature of such steam is 338° F. Let us admit this steam to the cylinder during a portion of the stroke, and cut it off at such a point that it will expand down to 50 pounds absolute pressure by the time the piston has reached the end of its stroke and the exhaust-valve opens. The temperature of steam at 50 pounds absolute pressure is 281° F., and the cylinder itself, or its inside walls, is at about the same temperature. Now, the exhaust-valve opens and most of the steam rushes out, leaving, say, 3 pounds back-pressure above the atmosphere due to the resistance offered by the valve opening and port. This 3 pounds is equal to 18 pounds absolute pressure, the temperature being 222° F. The temperature of the cylinder walls would, therefore, drop again, and would approach this temperature, 222° F. There would not be time enough for it to come clear down before steam is admitted on the other side of the piston for the return stroke, but we may use this illustration for our present purpose.

We have now the range of temperature before spoken of; that is, from 338° to 222°, or 116°, and it will be readily seen that this represents the possible fall of temperature of the cylinder during a stroke. But, now, more steam is admitted to the cylinder on the other side of the piston, and this is of the highest temperature; that is, 338°. The result is that some of it is condensed by coming in contact with the cylinder walls, which have cooled down, as we have shown. This is called *initial condensation;* it is a loss of power, and is, therefore, very objectionable.

2467. Let us now take the same conditions, but run our engine as a condensing engine, and see what the result will be. The steam enters the cylinder, as before, at a temperature of 338°. It is cut off somewhat earlier, if we wish to obtain the same power from the engine, and expands to the end of the stroke. Here the exhaust-valve opens, not to the atmosphere, but to the condenser, and the steam rushes out and is condensed, forming a partial vacuum. This vacuum may be equal to 26 inches of mercury, which is equivalent to 2 pounds absolute back-pressure, whereas before we had 18 pounds. The temperature of 2 pounds absolute pressure is 126° F., and our range of temperature becomes $338° - 126° = 212°$, as compared with the 116° of the first case. We should, therefore, have greater condensation of the steam when it is admitted for the next stroke. To prevent this, then, cross-compound engines are used. The steam is expanded partly in one cylinder, then transferred to the other, where it is expanded further, and is exhausted to the condenser. This obviates the necessity of admitting steam at a temperature of 338° F. into a cylinder which has just contained steam at a temperature of 126° F.

We assume, therefore, that our condensing engine is to be a cross-compound engine, and, as we have said, the hoisting-service requires two things of it: First, that it shall take live steam into the low-pressure cylinder when necessary, and, second, that its condenser and air-pump be of the independent type.

2468. A hoisting-engine which is to be condensing must take live steam into its low-pressure cylinder for the following reason: In the regular running of a cross-compound condensing engine, the steam is admitted from the boiler into the first or high-pressure cylinder. Here it is allowed to expand a certain amount, after which it is exhausted into a receiver. From the receiver it is admitted to the low-pressure cylinder, where it expands still further, and from there it is exhausted into the condenser. In other words, the low-pressure cylinder is fed from the high-pressure cylinder. Now, hoisting is not regular running; starting and stopping make up a large part of it. As has been pointed out, an engine capable of starting under load at any point must have no dead-center, but must have two cylinders, either of which is able to pick up the load when the other is on its center. But we have just seen that the low-pressure cylinder of a compound condensing engine is fed from the high-pressure cylinder; so, if the engine has been standing and no steam has been in the high-pressure cylinder, there will be none for the low-pressure, unless other provision be made for it. Also, the low-pressure cylinder may be the one required to pick up the load in starting. Therefore, to give it power at such a time, live steam is admitted to it independently of the high-pressure cylinder. This is simply done by having an *auxiliary steam-pipe* and *throttle-valve* leading directly to it, and arranged so that the engineer can open and shut it at pleasure. The steam used is generally reduced in pressure from that carried in the boiler by a *regulator*, because the high-pressure steam would give too much power, the low-pressure cylinder being of large diameter, so that it can do its part of the work with the partially expanded steam from the high-pressure cylinder.

2469. A hoisting-engine to run condensing should also have an *independent air-pump and condenser*. The air-pump of an ordinary condensing engine is operated by the engine itself and is practically part of it. If this plan is adopted in the case of a hoisting-engine, when the engine stops at

the end of a hoist the air-pump will stop, and the vacuum which it keeps up will be lost while the engine is standing. That is to say, atmospheric pressure will find its way into the condenser and act as a back-pressure against the low-pressure piston, amounting to about 15 lb. per square inch. This will, in most cases, render the low-pressure piston unable to pick up the load at the beginning of the next hoist. Independent condensers and air-pumps can be bought ready-made to suit any engine. They are simply special steam-driven pumps.

SIZE OF ENGINE.

2470. By the size of an engine is meant the diameter of its cylinder (or cylinders) and the length of its stroke. We should be able to say what these dimensions should be for a given case. They, of course, depend upon the work to be done, and this work consists, in the case of a hoisting-engine, of three things; namely, lifting the load, accelerating the moving parts, and overcoming the friction. How accurately these three items should be considered depends upon the circumstances of each case, but it is not generally advisable to figure too close. In fact, it is always advisable to have an excess of power in the engine. A little extra power costs very little if put into the engine while building, but it may be very difficult and costly to obtain it at some later day. Furthermore, the conditions are so variable that no rule can be laid down that will be applicable in all cases. It is, therefore, thought advisable to show the student how to work out a calculation approximately and to illustrate it with examples.

2471. Suppose that we wish to build a winding-engine for a shaft or vertical hoistway, the depth of which is 1,500 ft. The weight of material to be hoisted at each trip is 4,000 lb.; the weight of the mine-car to be used is 2,000 lb.; the weight of the cage is 3,000 lb. The shaft is to be double, with two cages balancing each other, and a tail or balance rope is to be used. The engine is to be of the duplex

type, direct-acting, and the mean effective pressure is to be 45 lb. per square inch.

The load on the rope is as follows:

Weight of material	4,000 lb.
Weight of mine-car	2,000 lb.
Weight of cage	3,000 lb.
Weight of rope, say	4,000 lb.
Total	13,000 lb.

The weight of the rope here used is assumed in order to get at the probable total weight on it. We will use a plow-steel wire rope, and a factor of safety of 10. The breaking strength of the rope should then be 13,000 lb. × 10, or 130,000 lb. By referring to Table 46, we find that a plow-steel wire rope having 19 wires to the strand, with a breaking strength of 130,000 lb., or 65 tons, slightly exceeds $1\frac{1}{8}$ in. diameter; but, as we have used a large factor of safety, the $1\frac{1}{8}$-in. rope, with a breaking strength of 60 tons, will be strong enough. This rope weighs 2 lb. per foot of length, or 2 lb. × 1,500 = 3,000 lb. Revising our figures above by using this corrected weight of rope, we have

Weight of material	4,000 lb.
Weight of mine-car	2,000 lb.
Weight of cage	3,000 lb.
Weight of rope	3,000 lb.
Total	12,000 lb.

The total load on the rope is, therefore, actually 6 tons, which, divided into 60 tons, the breaking strength of the rope, gives us a factor of safety of 10, as first assumed.

The diameter of the drum on which a rope containing 19 wires to the strand is to be wound should not be less than 60 times the diameter of the rope. Therefore, the minimum size of drum to be used with a $1\frac{1}{8}$-in. plow-steel rope is $5\frac{5}{8}$ ft. in diameter. Suppose, however, that we are not limited as to space; we will use a drum 8 ft. in diameter, because it is easier on the rope, and for a given hoist a larger drum need not be so long; consequently, the engines will not have to be spread so far apart, and the fleeting of the rope will not be so great.

We are now ready to calculate the work to be done. From the conditions laid down in the beginning, we note that the weight of the rope is balanced by the use of a tail-rope; that the two cages and the two cars balance each other, and that we have only the weight of the material, 4,000 pounds, as a net load, or, in other words, an unbalanced load.

To this we will add, for accelerating the moving parts and overcoming the friction, 10% of the gross load to get the actual load. By the gross load we mean all of the moving parts; that is:

1 lot of material	4,000 lb.
2 mine-cars	4,000 lb.
2 cages	6,000 lb.
2 ropes	6,000 lb.
Total	20,000 lb.

2472. We take the gross load for this purpose, because it is more nearly proportional to the friction and the inertia than the net load is. For instance, suppose in one case, which we will call Case A, we have conditions as laid down in our example; that is, a double shaft with two cages and two cars balancing each other, and hoisting 4,000 lb. of material at a hoist, with a tail-rope to balance the main rope. The gross load would be 20,000 lb. as above, and the net load would be 4,000 lb. Then, suppose in another case, which we will call Case B, we have the same conditions, except that no tail-rope is to be used. The gross load for this case would be:

1 lot of material	4,000 lb.
2 mine-cars	4,000 lb.
2 cages	6,000 lb.
1 rope	3,000 lb.
Total	17,000 lb.

And the net load would be:

1 lot of material	4,000 lb.
1 rope	3,000 lb.
Total	7,000 lb.

Now, it is quite evident that there will be less friction and inertia to overcome in Case B than in Case A, because the mass to be handled is less by the amount of one rope. We have just seen that the gross load of Case B is less than that of Case A by this same amount; therefore, the gross load, the friction, and the inertia are proportional. The net load of Case B, on the other hand, is greater than that of Case A, so it is not proportional to the friction and inertia.

Let us suppose still another case, which we will call Case C, in which we have the same weight of rope, cage, car, and material to be hoisted, but at a single shaft, where there is room for only one cage and car. We will not have, in this case, any balancing of the cage, car, or rope, and our gross load becomes:

1 lot of material	4,000 lb.
1 mine-car	2,000 lb.
1 cage	3,000 lb.
1 rope	3,000 lb.
Total	12,000 lb.

And our net load is the same. The mass to be handled in this case is less than that in Case A by the amount of one rope, one cage, and one car; that is, the two amounts are to each other as 20 is to 12, and it will be seen, on reflection, that the friction and inertia must also be to each other as 20 is to 12, for Cases A and C. Now the net load in Case C is 12,000 pounds, or three times as much as that in Case A, so that if we proportioned our work for overcoming the friction and accelerating the moving parts according to the net load, we would have three times as much for Case C as for Case A, while the mass to be moved is only twelve-twentieths as much.

2473. This method of calculating the work necessary to overcome the friction and to accelerate the moving parts is, in most cases, entirely satisfactory, although sometimes a larger percentage is used if the work is to be of a rough character, giving greater friction, or if great speed of

hoisting is to be required, thereby calling for a greater accelerating force. The amount of friction is at best a matter of judgment and not of calculation.

To calculate the work of acceleration, so much must be assumed that it is generally as satisfactory to assume the work directly. Furthermore, as we have seen, it is necessary to figure on one cylinder to do the total work, because the second one is at times powerless, and this gives us some extra power when both cylinders are acting.

As has been said, 10% of the gross load will be added to the net load to cover the work of overcoming the friction and accelerating the moving parts.

Rule.—*To find the actual load on the engines, add to the net load 10% of the gross load.*

The actual load for the original case will then be 4,000 pounds plus 10% of 20,000 pounds; that is, 6,000 pounds.

2474. The diameter of the drum is 8 feet, and to this must be added the diameter of the rope, $1\frac{1}{8}$ inches, to give the working diameter, which is, therefore, 8.094 feet, nearly.

The working circumference is, then, $8.094 \times 3.1416 = 25.43$ ft., nearly. For every revolution of the drum, we require $25.43 \times 6,000 = 152,580$ ft.-lb. of work from the engine.

Rule.—*To find the work required of the engines per revolution of the drum, multiply the actual load in pounds by the working circumference of the drum in feet.*

2475. The original proposition calls for a duplex direct-acting engine. This makes the revolutions of the engine and drum equal. Furthermore, we must count on only one engine to do the work.

The work performed in the cylinder of an engine per revolution may be calculated as in Art. **2067**, *Steam-Engines.* The force is the mean effective pressure P multiplied by the area A of cylinder in square inches. The distance moved by the piston per revolution is $2L$ inches, or $\frac{2L}{12}$ feet, where

L denotes the length of stroke in inches. Using the symbols of the article just referred to and letting w represent the work per revolution, we have

$$w = \frac{2 P L A}{12} = \frac{P L A}{6}.$$

For the present purpose, the formula must be modified, because we do not wish to calculate the work that an engine can do, but wish to calculate the size of an engine that can do the work required at the drum. The formula may be changed to read thus:

$$A L = \frac{6 w}{P}.$$

That is, *the area of the piston multiplied by the stroke in inches is equal to six times the work divided by the mean effective pressure.* The area of the piston is equal to .7854 times the square of the diameter, or $A = .7854 D^2$ where D is the diameter. Let the stroke be r times the diameter; that is, $L = r D$. If we now put these values of A and L in the formula, it becomes

$$.7854 D^2 \times r D = \frac{6 w}{P},$$

or

$$.7854 r D^3 = \frac{6 w}{P}.$$

This formula can be changed so as to read

$$D = 1.97 \sqrt[3]{\frac{w}{P r}}. \qquad \textbf{(211.)}$$

EXAMPLE.—What should be the size of the cylinders of a hoisting-engine which is to perform 152,580 ft.-lb. of work per revolution, if the mean effective pressure is 45 pounds per square inch and the stroke of the piston is twice its diameter?

SOLUTION.—In this case $r = 2$, since, if the stroke is twice the diameter, $r = \frac{2 d}{d} = 2$.

Applying formula **211**,

$$D = 1.97 \sqrt[3]{\frac{w}{P r}} = 1.97 \sqrt[3]{\frac{152,580}{45 \times 2}} = 23.5 \text{ in.}$$

The engine should, therefore, be a duplex engine with cylinders $23\frac{1}{2}$ in. in diameter and $23\frac{1}{2} \times 2 = 47$ in. stroke. Ans.

An engine of this size will do the hoisting at a vertical shaft 1,500 ft. deep, where the weight of material to be hoisted at a trip is 4,000 lb.; the weight of the mine-car to be used is 2,000 lb., and the weight of the cage is 3,000 lb.; two cages and a tail-rope being used, and the mean effective pressure being 45 lb. per sq. in.

2476. Before applying the formula to another example, the correctness of the above cylinder sizes may be tested in the following manner: Under the head of Duplex Engines, it was seen that when one piston is at the end of its stroke, and its crank is, therefore, on a dead-center, the other piston may be called upon to lift the maximum load. In such a case, this other piston is at about mid-stroke and its crank is at right angles to the connecting-rod, or in its most effective position. Now, the ability of an engine to rotate a drum is measured by the turning moment that it can exert, and the engine in the above position can exert a turning moment as follows:

The area of the piston is $23.5^2 \times .7854 = 433.7$ sq. in., which, multiplied by the mean effective pressure, 45 lb. per sq. in., gives 19,516.5 lb. as the total pressure exerted by the piston. This pressure is transmitted through the connecting-rod to the crank-pin, and acts on the latter at right angles to the crank. The length of the crank is half the length of the stroke, that is, $23\frac{1}{2}$ inches, and this, multiplied by the acting pressure given above, is the turning moment in inch-pounds that the engine will exert when at one of its dead-centers. This turning moment is, therefore,

$$19{,}516.5 \times 23\tfrac{1}{2} = 458{,}638 \text{ in.-lb.}$$

The resistance that this turning moment must overcome is the opposing turning moment due to the actual load on the drum. In the foregoing example this actual load is 6,000 lb. The working diameter of the drum is 8.09 ft., nearly; its radius is 4.05 ft., nearly, or 48.6 in., and the resulting turning moment is 48.6 in. $\times$ 6,000 lb. = 291,600 in.-lb.

It will be seen that this moment is considerably smaller

than that which the engine exerts, so the cylinder is of ample size to start the load.

2477. Consider now another example, similar to the foregoing, but differing from it in some of its conditions. Suppose, for instance, that the only difference in the two cases is that a tail-rope can not be used in the present case. The load on the rope will be the same as before, and we will, therefore, use a $1\frac{1}{8}$-in. plow-steel rope, which gives us a factor of safety of 10. Of course, the load on the rope will not be uniform, and it is the maximum load that has been taken; but, then, it is the maximum load that must be counted on. Naturally, also, the same diameter of drum will be used, namely, 8 feet.

Now, the conditions are that the shaft is double, and consequently, as before, the two cages and the two cars balance each other; but the net load is the weight of the material, 4,000 lb., and that of the rope, 3,000 lb., or a total of 7,000 lb. To this is added, as before, for accelerating the moving parts and overcoming the friction, 10% of the gross load, which is in this case

1 lot of material................	4,000 lb.
2 mine-cars.....................	4,000 lb.
2 cages.........................	6,000 lb.
1 rope..........................	3,000 lb.
Total	17,000 lb.

Ten per cent. of this is 1,700 lb., and the actual load to be calculated for is 7,000 lb. + 1,700 lb., or 8,700 lb.

The working circumference of the drum is, as before, 25.43 ft., and for every revolution is required $8{,}700 \times 25.43 = 221{,}241$ ft.-lb. of work from one engine. We will use the same style of engine, that is, a duplex engine and direct-acting; so this is the amount of power that is necessary per revolution of one engine. Then, if the same proportion of cylinders is adopted, formula **211** gives

$$D = 1.97 \sqrt[3]{\frac{221{,}241}{45 \times 2}} = 26.59 \text{ in., say } 26\tfrac{1}{2} \text{ in.}$$

Stroke $= 26.5 \times 2 = 53$ in.

That is to say, the engine should have cylinders $26\frac{1}{2}$ inches in diameter, with a 53-inch stroke.

It will be observed that this engine is considerably larger than the first ones, yet it does only the same amount of useful work. It hoists the same weight of material, 4,000 lb. at each hoist, but it also lifts the entire weight of the rope at starting when the loaded car is at the bottom of the shaft. The work of lifting the rope is so much energy thrown away. This loss is offset in some degree by the wear and tear of a tail-rope.

2478. Let us carry this subject one step further and see what size of engine we would require if it were necessary to hoist the same material from a single shaft, that is, the same amount at one hoist. The load on the rope would be the same as it was in the first case:

Weight of material............	4,000 lb.
Weight of mine-car............	2,000 lb.
Weight of cage................	3,000 lb.
Weight of rope................	3,000 lb.
Total.................	12,000 lb.

We shall, therefore, use the same rope, a $1\frac{1}{8}$-inch plow-steel rope, and the same drum, 8 ft. in diameter. As the shaft is single, there is only one cage and one car, so these are not balanced, and the net load becomes the same as the above load on the rope, that is, the same as the gross load, 12,000 lb. Increasing this by 10% to cover the work of acceleration and overcoming the friction, we have 13,200 lb. for the actual load. Multiplying this by 25.43 feet, the working circumference of the drum, gives 335,676 ft.-lb. of work to be done per revolution.

Using formula **211**,

$$D = 1.97\sqrt[3]{\frac{335,676}{45 \times 2}} = 30.55 \text{ in., say } 30\tfrac{1}{2} \text{ in.}$$

$$\text{Stroke} = 30.5 \times 2 = 61 \text{ in.}$$

In other words, the engine must have cylinders $30\frac{1}{2}$ inches in diameter, with 61-inch stroke.

A study of the preceding examples is instructive. An engine with 30½-inch by 61-inch cylinders hoists only the same weight of material as the first engine did, with 23½-inch by 47-inch cylinders. Furthermore, it can make only half as many hoists in a given time as the first engine can, because it loses all the time during which it is sending the cage down for another load, whereas in the first and second cases a hoist is made during that time. In this third case, the work necessary to lift the rope, cage, and car is all useless work, and is so much thrown away. These are certainly great disadvantages, but there are some advantages belonging to this last system. There is only one drum, one rope, and one cage to supply and keep in repair, and the cost of sinking a single shaft is, of course, much less than that of sinking a double shaft.

In many mining operations there is great uncertainty as to the permanency of the mine, because so little is known about the hidden treasures that are sought. In such a case, it is often advisable to sink the smaller shaft and endure the extravagance of hoisting without a balance.

2479. Suppose that, instead of being direct-acting, the engine is on the second motion. It is evident that the size of the cylinders can be reduced, although the horsepower will remain the same, owing to the increase in the number of revolutions. Thus, if the ratio of the gear to the pinion is 3 : 1, that is, if it takes three revolutions of the pinion to turn the drum once, the engine will make 6 strokes per revolution of the drum instead of 2, as in the previous cases.

If we represent by r_1 the ratio of the gear to the pinion, in other words, if the diameter of the gear is r_1 times that of the pinion, formula **211** may be altered slightly to apply to second-motion engines. It then becomes

$$D = 1.97\sqrt[3]{\frac{w}{P r r_1}}. \qquad \textbf{(212.)}$$

Formula **212** may be expressed in words as follows:

The diameter of the cylinder of a second-motion engine is

equal to 1.97 times the cube root of the quotient obtained by dividing the work in foot-pounds per revolution of drum by the continued product of the M. E. P. in pounds per square inch, the ratio of the stroke to the diameter, and the ratio of the gear to the pinion.

EXAMPLE.—Suppose that the work to be done per revolution of drum is 152,580 ft.-lb., that the stroke is $1\frac{3}{4}$ times the diameter, and that the gear has $3\frac{1}{2}$ times as many teeth as the pinion; what should be the size of the cylinders, the M. E. P. being 45 lb. per sq. in.?

SOLUTION.—In this case $r = 1\frac{3}{4} = 1.75$, and $r_1 = 3\frac{1}{2} = 3.5$.

Applying formula **212**,

$$D = 1.97\sqrt[3]{\frac{152,580}{45 \times 1.75 \times 3.5}} = 16.18 \text{ in., say } 16 \text{ in. Ans.}$$

Then, $L = 16 \times 1\frac{3}{4} = 28$ in. Ans.

2480. Steam and Compressed-Air Engines.—So far, nothing has been said about the kind of power utilized by the engine. It may be either steam or compressed air. In most cases, where the engine is located on the surface, steam is preferable, because it can be obtained directly from the boiler without the intervention of an air-compressor and the consequent loss of total efficiency; but this is not always so. Consider the case of a mining plant which has compressors for supplying air for rock-drills, and where it becomes necessary to sink a shaft at a distance from the main works, and at a point where it would be very difficult to take fuel and water. Steam carried a long distance in pipes loses much by condensation; and, furthermore, compressed air must be carried to the new shaft for the rock-drills. It is quite evident, under such circumstances, that the engine should be supplied with compressed air instead of steam.

Again, it is often necessary to have a shaft or slope entirely underground; that is, to have the hoisting-engine and drum placed there, too. In such a location, steam would almost always be barred out on account of the heat and moisture that would be liberated with the exhaust. On the other hand, the exhausted air from an engine driven by compressed air would be cool and dry, and would supply fresh

air to the miners. Hoisting-engines may then be driven by steam or compressed air, and it would be well to know which is to be used when deciding upon the engine.

2481. Throttle-Valves.—The throttle-valve of a hoisting-engine, whether the engine is designed to be run by steam or compressed air, should be balanced, so that it can be easily opened or shut by hand; and it should be of the lever type, so that it can be opened or shut quickly. There should also be a supplementary valve close to the throttle and easy of access, which can be shut in case of any trouble with the throttle. This may be of the screw type, should be absolutely tight, and should be closed whenever the engines are to stand for a considerable length of time.

2482. Cylinder Relief-Valves.—In any steam-engine some of the steam that is admitted to the cylinder to do the work is condensed, partly by the absorption of its heat by the cylinder walls, and partly by the conversion of its heat into work. This produces water, which must always be taken care of. Let us for a moment follow out the operation of an engine. The piston is at one end of the cylinder. Steam is admitted behind it, and forces it along the cylinder to the other end, condensing somewhat during that time, and leaving the cylinder full of steam (at a lower pressure) and water. The return stroke now takes place, and this steam is let out through the exhaust-valve, while the water is pushed along the bottom of the cylinder, gradually increasing in depth as its area is decreased, until it reaches such a height that it can also get out through the exhaust-valve. It can not get out quickly enough, however, and towards the end of the stroke the exhaust-valve is closed, thus shutting in the water that is left. Now, if this amount is more than enough to fill the clearance space, that is, the space between the piston at the end of its stroke and the valve, trouble will follow, because the piston must go on to the end of its stroke, being controlled by the revolution of the crank and the inertia of all the moving parts of the

engine. As water is non-compressible, the result in such a case would probably be the blowing out of the cylinder head.

Now, with a hoisting-engine, where the running is intermittent, the cylinders have time to cool down considerably during the stoppages, and there is, consequently, much condensation at the beginning of each hoist. To take care of this water of condensation, therefore, all hoisting-engines should be provided with some sort of relief-valves. On small engines it has been found sufficient to have drip-cocks at the bottom of each end of each cylinder, so arranged that they can be opened and closed readily by the engineer; but with larger engines it has often been found necessary to have valves of larger area held shut by a spring, and arranged to be opened automatically by any excessive pressure in the cylinder.

2483. A very excellent device for this purpose is a combined *relief-valve*, such as is shown in Fig. 892, or some modification of it. The idea here is to have a small valve *A*

FIG 892

opening in towards the cylinder and held open by a spring, so that any water that is in the cylinder at the end of the hoist, or that would accumulate in the cylinder from leaky valves during a stoppage, can drain away. When the steam is admitted to the cylinder to make a hoist, it shuts this valve

and holds it shut by its pressure. Now, it will be noticed that this valve seats on another valve *B*, which opens out from the cylinder, but which is held shut by a heavy spring against any ordinary pressure in the cylinder. In case of any excessive pressure due to water in the cylinder, or other causes, this larger valve will be forced open, and thus relieve the pressure.

DRUMS.

2484. Having considered the motors available for a hoisting plant, the next subject requiring attention is that of the drums. These are of four kinds, which will be taken up separately. They are as follows: *Cylindrical Drums; Conical Drums; Reels; Rope Wheels.*

CYLINDRICAL DRUMS.

2485. Cylindrical drums probably form the largest class of drums in use. Fig. 888 shows a cylindrical drum of the simplest type. This is a plain iron cylinder with a flange at each end to prevent the rope from running off. It is designed for a hemp or manilla rope and for a short hoist. Its diameter is too small to allow an iron or steel rope to be wound upon it, and it has not the capacity to wind any considerable length of rope, although in a case like this it would be permissible to allow the rope to wind upon itself after once filling the drum. Fig. 889 shows a similar drum of larger diameter. Such a drum is also only applicable to a short hoist, although it would be large enough to wind the small sizes of iron and steel ropes. In Figs. 890 and 891 are shown two cylindrical drums, essentially alike, built for larger and heavier service. These are each of cast iron, and each has a spiral groove cast or turned in it for the rope to lie in as it is wound. In such cases, and, in fact, whenever metal ropes are used, it is not good practice to allow the rope to wind upon itself, because the different coils wear upon each other, although this is sometimes done. The drum should be built large enough to take the full length of the rope

required by the hoist, running over the drum only once. For example, suppose we are required to make a hoist of 1,000 feet and to lift a load which would require a $1\frac{1}{4}$-inch steel rope. The diameter of the smallest drum which it would be advisable to use is $1\frac{1}{4}$ in. $\times$ 60 = $6\frac{1}{4}$ ft., and the diameter at the center of the rope when wound upon the drum is 6 ft. $4\frac{1}{4}$ in. The corresponding circumference, or the length of one coil of rope, is nearly 20 ft. To wind 1,000 ft. would require 1,000 $\div$ 20 = 50 turns on the drum. To this should be added at least one turn, say one and a half turns, at the end of the rope, to afford friction, so that all the strain will not come on the fastening; and about three turns should also be added for possible overwinding. This makes $54\frac{1}{2}$ turns to be allowed for on the drum. If the drum is of iron with grooves turned in it, we must allow $\frac{1}{4}$ inch between adjacent parts of the rope, or $1\frac{1}{2}$ inches from the center of one turn to the center of the next turn. This gives $54\frac{1}{2} \times 1\frac{1}{2}$ inches = $81\frac{3}{4}$ inches, or 6 feet $9\frac{3}{4}$ inches for the length of the drum between the flanges.

2486. The drums shown in Figs. 890 and 891 are made of cast iron, in one piece, and are of the design shown in Fig. 893. This makes a very good drum for small sizes.

The smaller sizes of drums, such as have just been considered, are also often made of wooden lagging carried on

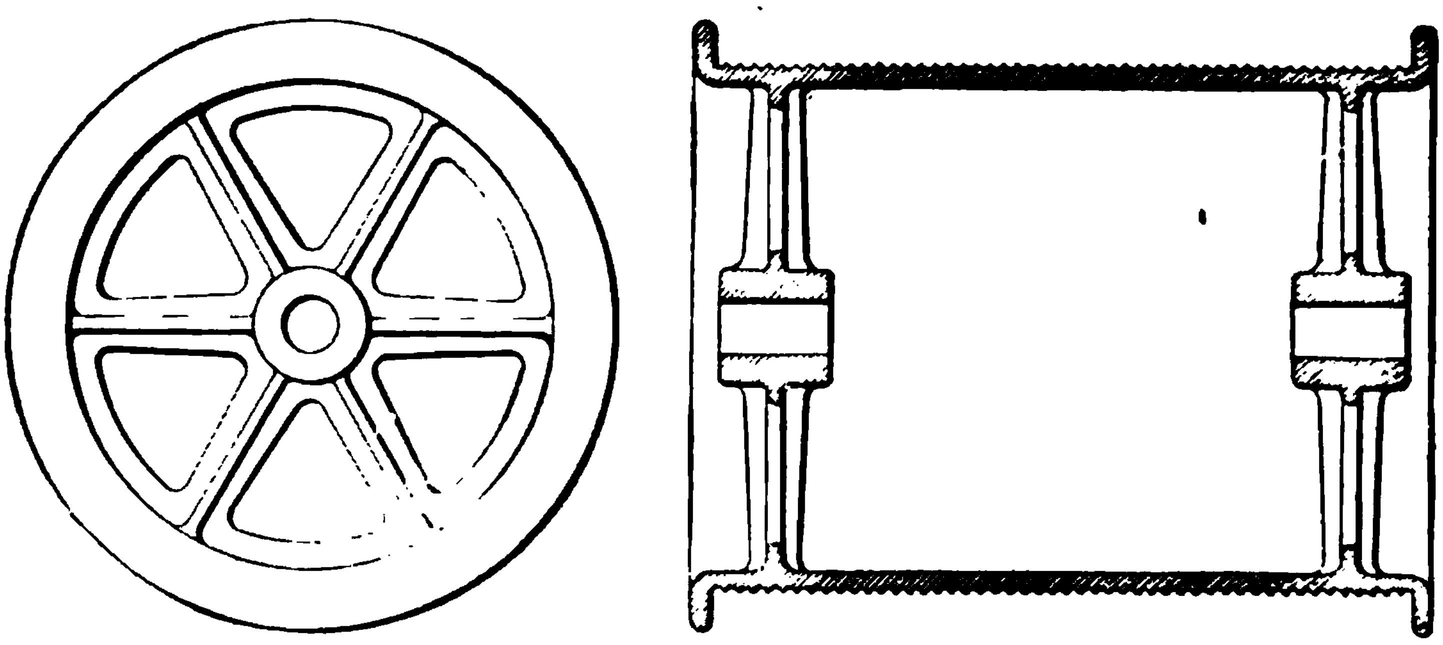


FIG. 893.

cast-iron spiders. In this case it is not necessary to allow the $\frac{1}{4}$ inch clearance between the coils of the rope. It can wind against itself, and so take up only $1\frac{1}{4}$ inches. The drum

would then need to be $54\frac{1}{2} \times 1\frac{1}{4}$ inches $= 68\frac{1}{8}$ inches, or 5 feet $8\frac{1}{8}$ inches long. Such a drum as this is shown in Fig. 894. It is intended for a direct-acting hoisting-engine, A being both the drum shaft and the crank-shaft. B and B' are the journals, and C and C' are the ends to which the cranks are fastened. Two kinds of cast-iron spiders D and D' are shown, one with a flange and one without. The spider with a flange is better than the other, but it costs more, and if a flange is not used, extra length must be added to insure that the rope shall not run off the end. In very long drums of this style, it is found necessary to add a third spider midway between the other two to stiffen the drum against collapsing.

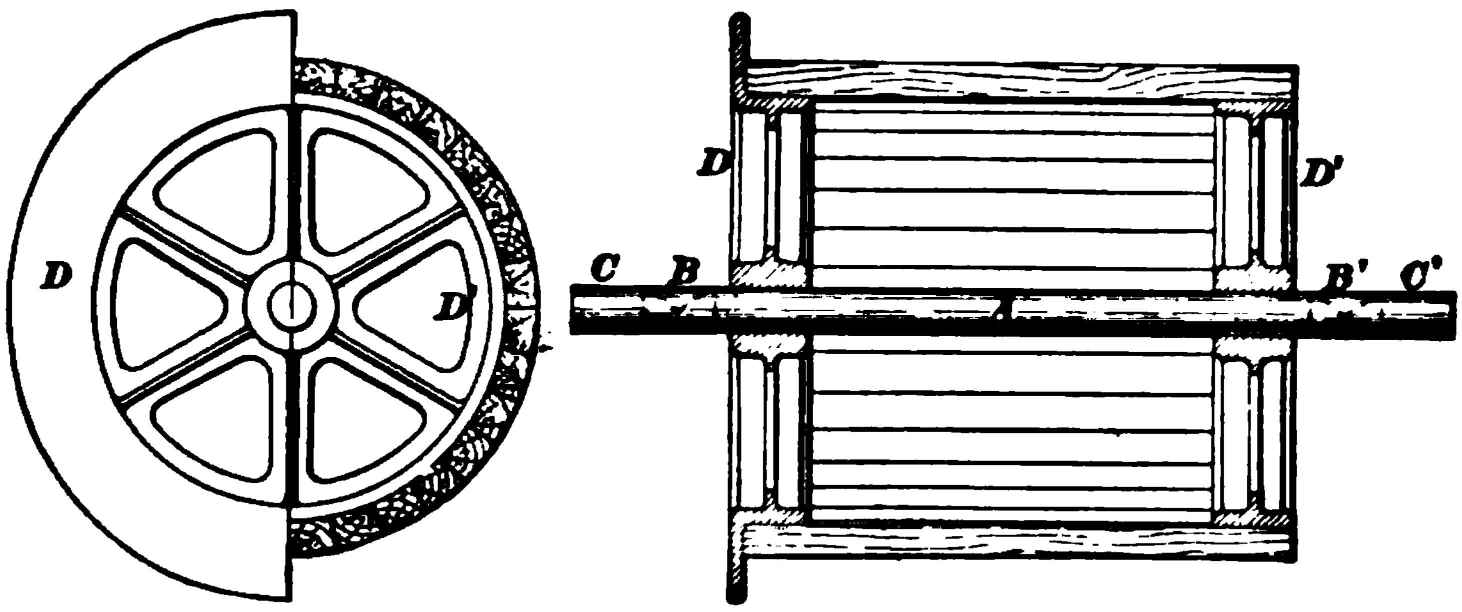


FIG. 894.

The lagging is bolted to the spiders, and the bolt-heads should be countersunk into it so as to clear the rope after it has bedded itself into the wood.

2487. Larger drums than the foregoing are often necessary, in order to wind larger ropes and greater lengths of rope. They are found as large as 30 feet in diameter and 20 feet long. Such drums are necessarily built up of several pieces. There is the hub, or sleeve, to go on the shaft, made either in one or two pieces, and with or without arms. Then there is the rim, made up of four, six, or eight segments, bolted together to form a cylinder. And, finally, there are the arms. These are made of cast iron, and designed to withstand compression, or of wrought iron, and designed to withstand tension. The latter method makes a lighter drum, though possibly a more expensive one.

CONICAL DRUMS.

2488. The **conical drum** is similar to the cylindrical drum, except in the form of its winding surface, which is in the shape of a frustum of a cone instead of in the shape of a cylinder. Conical drums are designed to take the place of cylindrical drums when it is necessary or advisable to equalize the load on the engines due to the weight of the rope. Suppose we have a single-compartment vertical shaft 800 ft. deep; that we are required to hoist 5,000 lb. of material at a trip; that the weight of the mine-car to be used is 3,000 lb.; and that the weight of the cage is 3,000 lb.

The load on the rope would then be:

Weight of material............	5,000 pounds.
Weight of mine-car............	3,000 pounds.
Weight of cage................	3,000 pounds.
Weight of rope, say...........	3,000 pounds.
Total................	14,000 pounds.

Let us use a cast-steel rope having 19 wires to a strand and a factor of safety of about 10. The breaking strength of the rope should then be about 14,000 lb. × 10, or 140,000 lb. By consulting Table 46, it is seen that a $1\frac{3}{8}$-inch rope has a breaking strength of 63 tons, or 126,000 lb., and that 800 ft. of it weighs 2,400 lb. Using this corrected weight to sum up the load on the rope, we have, instead of the above,

Weight of material............	5,000 pounds.
Weight of mine-car............	3,000 pounds.
Weight of cage................	3,000 pounds.
Weight of rope................	2,400 pounds.
Total................	13,400 pounds.

The breaking strength divided by this total load gives 9.4 as the factor of safety, which is sufficient. Now, the minimum diameter of a drum on which a $1\frac{3}{8}$-inch rope, having 19 wires to the strand, should be wound is 60 times the diameter of the rope, or $1\frac{3}{8}$ in. $\times 60 = 82\frac{1}{2}$ in., or nearly 7 ft.

Let us then decide upon a drum 7 ft. in diameter, and we have the necessary data to go on with our calculation. At the beginning of the hoist, when the cage is at the bottom and has a loaded car on it, the load is 13,400 pounds, as above. This load is supported at the circumference of the drum, and if we multiply it by the radius of the drum, we will have a turning moment that must be overcome by the engine in order to make the hoist. This turning moment is 13,400 pounds × 3½ feet, or 46,900 foot-pounds. At the end of the hoist the load is 11,000 pounds.

That is,

Weight of material........	5,000 pounds.
Weight of mine-car........	3,000 pounds.
Weight of cage............	3,000 pounds.
Total	11,000 pounds.

Multiplying this by the radius of the drum, the turning moment is 11,000 pounds × 3½ feet, or 38,500 foot-pounds. From this it appears that with a cylindrical drum the load against the engine is much greater at the beginning of the hoist than it is at the end of the hoist.

2489. Let us now examine another case. Suppose we have a double-compartment vertical shaft of the same depth, and that we are to hoist the same amount of material at a trip, in the same mine-car and on the same cage; but that an empty car and cage will be lowered in one compartment while the loaded car and cage are hoisted in the other. The two cars will then balance each other, the two cages will balance each other, and the loads will be as follows: At the beginning of the hoist, when the loaded car and cage are at the bottom, the gross load is 13,400 pounds.

That is,

Weight of material.........	5,000 pounds.
Weight of mine-car	3,000 pounds.
Weight of cage	3,000 pounds.
Weight of rope	2,400 pounds.
Total.............	13,400 pounds.

Multiplying this by the radius of the drum, the gross turning moment is 13,400 pounds × 3½ feet, or 46,900 foot-pounds, as before, but there is a counterbalancing load of 6,000 pounds.

That is,

Weight of mine-car	3,000 pounds.
Weight of cage	3,000 pounds.
Total.............	6,000 pounds.

This means a counterbalancing turning moment of 6,000 pounds × 3½ feet, or 21,000 foot-pounds. The net turning moment to be overcome by the engine at the beginning of the hoist is, therefore, 46,900 − 21,000 = 25,900 foot-pounds.

At the end of the hoist there is a gross head of 11,000 pounds.

That is,

Weight of material........	5,000 pounds.
Weight of mine-car........	3,000 pounds.
Weight of cage	3,000 pounds.
Total	11,000 pounds.

This is equal to a gross turning moment of 11,000 pounds × 3½ feet, or 38,500 foot-pounds. We also have a counterbalancing load of 8,400 pounds.

That is,

Weight of mine-car	3,000 pounds.
Weight of cage	3,000 pounds.
Weight of rope	2,400 pounds.
Total	8,400 pounds.

This is equal to a counterbalancing turning moment of 8,400 pounds × 3½ feet, or 29,400 foot-pounds, and leaves a net turning moment against the engine of 38,500 − 29,400 = 9,100 foot-pounds. In other words, the turning moment that the engine has to supply varies from 25,900 foot-pounds at the beginning of the hoist to 9,100 foot-pounds at the end of the hoist.

2490. It is to equalize the two loads and to give the engine the same amount of work to do throughout the hoist

that we resort to the conical drum; for it will readily be seen that the great difference in the two loads figured is due to the weight of the rope, and that if the radius of the drum

FIG. 895. FIG. 896.

varies in the opposite direction to the variation in the load due to the rope it will eliminate this difference. To determine what these radii of the drum should be, we will refer to the accompanying diagrams, Figs. 895 and 896, which

represent the proposed hoist fitted with conical drums. In Fig. 895 we have the condition at the beginning of the hoist. Cage *B* is at the bottom and carries a loaded car; cage *T* is at the top and carries an empty car; and we are ready to make the hoist. The turning moment which the engine must overcome is equal to the weight of the material to be hoisted plus the weight of the cage and car at *B*, plus the weight of the rope, multiplied by the small radius of the drum, minus the weight of the car at *T*, multiplied by the large radius of the drum.

Suppose now that the hoist has been made and that we have, as shown in Fig. 896, the condition of things at the end of the hoist. The cage with the loaded car, which was at *B* and hanging on the small diameter of the drum, is now at the top at *L* and hanging on the large diameter. The cage with the empty car, which was at *T* and hanging on the large diameter of the drum, is now at the bottom at *E* and hanging on the small diameter. The turning moment which the engine must overcome is now equal to the weight of the material hoisted plus the weight of the cage and car at *L*, multiplied by the large radius of the drum, minus the weight of the cage and car at *E*, plus the weight of the rope, multiplied by the small radius of the drum.

2491. Now what is desired is that the load against the engine at the beginning of the hoist shall be equal to that at the end of the hoist. To determine what diameters of drum will produce such an effect, we have simply to make an equation of the two loads, and then solve the equation for the relation between the two diameters.

Let M = the weight of material;
C = the weight of cage and car;
R = the weight of rope;
D = the large diameter of drum;
d = the small diameter of drum.

Then, writing down the loads, as above, we get this equation:

$$(M + C + R)\,d - C \times D = (M + C)\,D - (C + R)\,d.$$

It will be noted that, instead of the radii of the drum, as used before, we have here used the diameters. That is because we generally speak of the diameter of a drum and not its radius, when referring to its size. In using the radius, we obtain the actual moment in foot-pounds, but in this case we do not care for the actual moment. We only want their relative values, and these can be obtained just as well by using the diameters for the multipliers.

To solve the above equation, we will proceed as follows:

Add $C \times D$ to both sides of the equation, and we have

$$(M + C + R)d = (M + C)D - (C + R)d + C \times D.$$

Add $(C + R)d$ to both sides, and we have

$$(M + C + R)d + (C + R)d = (M + C)D + C \times D,$$

which turned about is

$$(M + C)D + C \times D = (M + C + R)d + (C + R)d.$$

This may be written

$$D(M + 2C) = d(M + 2C + 2R).$$

Dividing both sides by $(M + 2C)$, we have

$$D = \frac{d(M + 2C + 2R)}{(M + 2C)}. \qquad \textbf{(213.)}$$

Rule.—*To find the large diameter of a conical drum, **multiply** the small diameter by the weight of the material to be hoisted, plus twice the weight of the cage and car, plus twice the weight of the rope; divide this product by the weight of the material, plus twice the weight of the cage and car.*

Applying this rule to the plant that we have been considering, we have

$$D = \frac{7(5,000 + 12,000 + 4,800)}{(5,000 + 12,000)} = 8.96 \text{ ft.}$$

The drum would then be 7 ft. in diameter at the small ends and 8 ft. 11½ in. at the large ends.

2492. Conical drums are more expensive to build than cylindrical drums, and they are not, therefore, used as

often. In capacity, they range as large as the largest cylindrical drums.

For hoisting from shafts less than 2,000 ft. deep, cylindrical and conical drums answer very well, but for hoisting from shafts of greater depths, they are not entirely satisfactory. As has been said, it is not good practice to wind the rope upon itself; and to wind it in a single layer on a drum, when there are three or four thousand feet of it, requires a very large drum. A very large drum is, of course, very heavy and costly. The great weight is objectionable because it forms a mass to be put into motion and brought to rest at each hoist, thus making the action slow or requiring surplus power in the engines for the purpose. It requires a large shaft and large bearings to carry it. The great length of drum necessitates the placing of the engines far apart, and this adds to the cost of the engines, foundations, and building. It also makes the fleeting of the rope, as it winds upon the drum from one end to the other, excessive, and this requires that the drum shall be placed at a considerable distance from the shaft and that carrying sheaves shall be used.

Because of these objections, other styles of drums have been tried with varying success. These are the *Reels* and the *Rope Wheels* before mentioned.

REELS.

2493. Reels are small narrow drums with high flanges, used for winding flat ropes. A drum of this style is illustrated in Fig. 897. The hub is increased in diameter above what is necessary for strength to such a size as is suitable to wind the rope upon. It is then cored out from the inside, so as not to leave too great a mass of metal. From the hub, arms of a **T** cross-section extend out radially to serve as a flange to support the rope laterally when it is all wound upon the reel. These arms are connected at their outer ends by a continuous flange having nearly an **L** cross-section, which is flared out so as to take the rope easily, if it is deflected sideways at all. The rope winds at first upon the

diameter $A\ A$ and then upon itself, so that the diameter of the reel increases as the hoist is made and as the load due to the rope decreases. This serves to equalize the load due

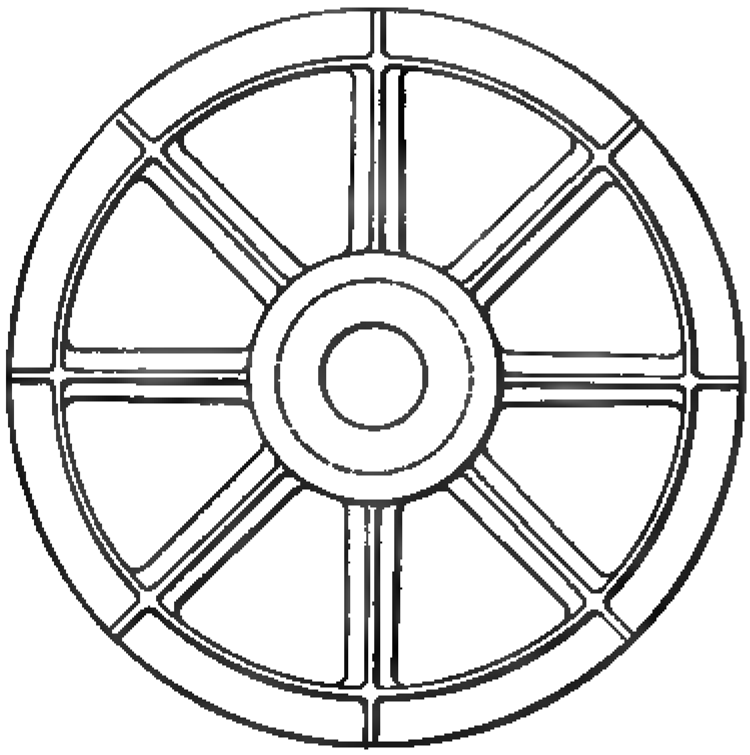


FIG. 897.

to the rope in the same manner as a conical drum does. Two reels are generally put upon the same shaft, and while one is hoisting from one compartment of a shaft, the other is lowering into another compartment.

2494. Let us take a case of this sort and calculate the diameters of the hub and flanges of the reel, and the size of the rope that should be used. Suppose, for instance, that we have a two-compartment vertical shaft 2,000 feet deep; that we are required to hoist 5,000 pounds of material at a trip, and that we are to hoist it in a self-dumping skip weighing 3,000 pounds.

The method of calculation will be somewhat similar to that used in determining the sizes of a conical drum, but not so direct, because, in this case, there is another variable quantity to contend with in the size of the rope. When using a round rope, we can tell directly by referring to a table what diameter of rope should be used for a given load; but, in using a flat rope, we can not tell by referring to the table what size of rope should be used, because we find that

several ropes give the desired strength; yet we do not know what thickness will suit the other conditions. We must, therefore, resort to the method of "trial and error" or use algebra. We will use the former. This will be clearer to the student as we proceed with the problem before us.

Referring to Table 49, Art. **2527,** we find that a flat steel rope with a breaking strength of 153,000 pounds weighs 5.15 pounds per foot; hence, 2,000 feet of it weigh 2,000 × 5.15 – 10,300 pounds. The total load on the rope would then be 18,300 pounds.

That is,

Weight of material........	5,000 pounds.
Weight of skip............	3,000 pounds.
Weight of rope	10,300 pounds.
Total.............	18,300 pounds.

This rope would give a factor of safety of 8.4, which is not quite enough when figuring from the dead load without that due to resistance of friction and acceleration.

A similar rope with a breaking strength of 204,000 pounds weighs 6.86 pounds per foot; hence, 2,000 feet of it weigh 2,000 × 6.86 = 13,720 pounds. The load on the rope would then be 21,720 pounds.

That is,

Weight of material........	5,000 pounds.
Weight of skip............	3,000 pounds.
Weight of rope	13,720 pounds.
Total.............	21,720 pounds.

This rope gives a factor of safety of 9.4, which is quite satisfactory.

Substituting the foregoing weights of material, skip, and rope in formula **213,** we have

$$D = \frac{d\,(5,000 + 6,000 + 27,440)}{(5,000 + 6,000)}.$$

$$D = 3.5\,d.$$

In other words, the large diameter, or that of the last coil of rope, should be 3.5 times the small diameter, or that of the

reel hub. If we assume the reel hub to be 4 feet in diameter, the last coil of the rope should be 3.5 × 4 feet, or 14 feet, in diameter, and we have such a coil of rope as is shown in Fig. 898.

The area of a circle 14 feet, or 168 inches, in diameter is 22,167 square inches, and the area of a circle 4 feet, or 48 inches, in diameter is 1,810 square inches. The difference

FIG. 898.

between these area, or 20,357 square inches, is the area of the annular ring which represents the rope. Now, the original proposition was that this rope should be 2,000 feet, or 24,000 inches, in length. If, then, we divide the area of the rope by its length, we will have its width, which in this case, of course, means its thickness. That is, 20,357 ÷ 24,000 = .84 inch, which is the thickness of a rope that would fulfil our requirements when wound upon a reel with a 4-foot hub. This is considerably thicker, however, than good practice would sanction, the thinner ropes, such as $\frac{3}{8}$ in., $\frac{1}{2}$ in., and $\frac{5}{8}$ in., being most in favor.

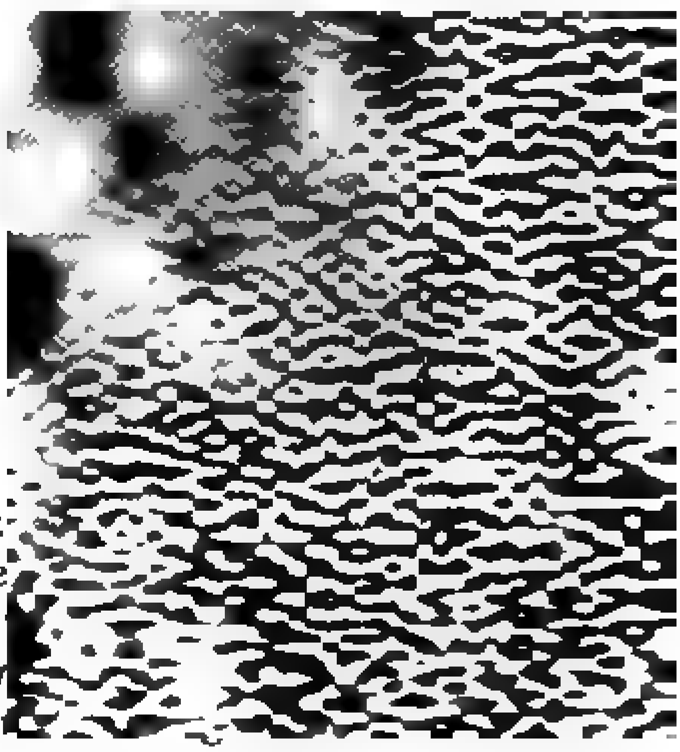
To obtain a thinner rope that will answer the same purpose, we will assume a reel hub 3 feet in diameter. The last coil of rope will then be 3.5×3 feet, or $10\frac{1}{2}$ feet, in diameter, and we have an annular ring of rope, as before. The area of a circle $10\frac{1}{2}$ feet, or 126 inches, in diameter is 12,469 square inches, and the area of a circle 3 feet, or 36 inches, in diameter is 1,018 square inches. The difference between these is 11,451 square inches, or the area of the rope, and this, divided by the required length, as before, gives .48 inch as the thickness of the rope. Call this $\frac{1}{2}$ inch. We find, by referring to Table 49, that a $\frac{1}{2}$-inch by 8-inch rope is the one we want. We then have for the dimensions of our reel—diameter of hub, 3 feet; width between flanges, $8\frac{1}{2}$ inches, allowing $\frac{1}{4}$ inch on each side of the rope for clearance; diameter of the flanges where they flare out, $10\frac{1}{2}$ feet.

2495. Reels have the advantage of being light and inexpensive to build. They are very narrow and require very short drum shafts, thus allowing the engines to come close together, and reducing the necessary foundations and building. They allow a partial equalization of the load on the engines, due to the rope, and they do not require any fleeting of the rope. These advantages, however, are more than counterbalanced by the disadvantages due to the use of a flat rope. These will be considered later in their proper place. Reels and flat ropes are used to some extent in the Western mining camps of the United States and in England. In the latter place they are no longer in favor among engineers, and several of them have been taken out to be replaced by drums using round ropes.

ROPE WHEELS.

2496. Rope wheels, as we speak of them here, are simply very short cylindrical drums, so short that only a few turns of the rope are accommodated. They are used for hoisting in two systems: *the Kœpe system* and *the Whiting system*, both of which use round ropes, and are designed to overcome the objections to cylindrical and conical drums.

2497. The Koepe System.—This may briefly be said to consist in the substitution of a single-grooved rope wheel in place of the ordinary drum. The winding-rope passes from one cage up over its head sheave; from there around the drum and back over the other head sheave, then down to the second cage. It simply encircles about half the periphery of the drum in the same manner as a driving-belt on an ordinary pulley, and is driven by the friction between the two. There is a balance rope beneath the cage, and the arrangement is, therefore, an endless rope, with the cages fixed at the proper points. The arrangement is shown in Fig. 899. The drum must, of course, be stronger



than t ing she to do usually wood i made to receive the winding-rope, the depth of this groove being generally equal to twice the diameter of the rope. Instead of being placed parallel, the head sheaves are placed at an angle with each other, pointing each to the groove in the drum. This

FIG. 899.

reduces the side friction of the rope on the sheaves. This

system has been in operation in Europe since 1877. Experiments made upon it have determined that, with a rope passing only one-half turn around the drum, the coefficient of adhesion is about 30%. This was with clean ropes. If the ropes are oiled, the adhesion becomes less, and slippage occurs, which, of course, is very objectionable. Slippage in such a case not only results in the wear of the drum lining, but it makes the reading of the hoist indicator incorrect, and leads to possible overwinding. Of course, if the hoist is indicated by marks on the rope, and if the engineer can see the cage itself, this is not so serious.

At the end of the hoist, if the upper cage is allowed to rest on the keeps, its weight and the weight of the balance rope are taken off from the hoisting rope. The consequence is that there is not enough weight left on the hoisting rope to produce sufficient friction with the drum to start the next hoist. To prevent this trouble, the keeps have been dispensed with, or else the rope has been made continuous and independent of the cage. To do this, cross-heads have been added above and below each cage, and connected by ropes or chains outside of the cages. The bridle chains are then hung from the top cross-head, and when the cage rests on the keeps the weight of the hoisting and balance ropes remains on the drum.

This system has its advantages and disadvantages. Only one drum is necessary for the operation of two compartments, and it is light and inexpensive to build. It is also very narrow, admitting of a short drum shaft and small foundations. The system permits a perfect balance of rope and cage, so that the work to be done by the engines is uniform, except for the acceleration, and consists only in lifting the material and overcoming the friction. There is no fleeting of the rope as it runs to and from the drum.

Yet with these points in its favor, it has disadvantages which prevent its being used to any considerable extent. Probably the greatest objection to the system is the liability of slippage of the rope on the drum. This, of course, must not occur, or there will be no end of trouble. Another

objection is that if the rope breaks both cages will surely go to the bottom. Still another, and a very important one, is that hoisting from different levels can not be done with entire satisfaction. This is because the cages are at a fixed distance from each other. The length of the rope, of course, is such that when one cage, which we will call *A*, is at the top, the other cage, which we will call *B*, is at the bottom. If hoisting is to be done from the bottom, this is satisfactory; but if hoisting is to be done from some upper level, cage *B*, which is at the bottom, must go up to that level and be loaded before it can go to the top. Then, when cage *B* goes to the top with its load, cage *A* must go clear down to the bottom, wait there while cage *B* is being unloaded, go up to the upper level and receive its load before it can start on its upward trip. For each trip, therefore, the time is lost that is necessary for a cage to go from the bottom to the upper level and be loaded; and two movements of the engines are necessary to make a hoist instead of one, as it should be. If, when cage *A* went to the top, cage *B* went down only to the desired level, and was being loaded there while cage *A* was being unloaded, it would be ready to go to the surface with its load directly, and this loss of time would not occur.

2498. The Whiting System.—This is a system of hoisting with round ropes, in which two narrow grooved drums or rope wheels are used in place of the cylindrical or conical drums ordinarily used. These drums are placed tandem. As shown in Fig. 900, the rope passes from one cage *A* up over a head sheave, down under a guide sheave, then to the drums *M* and *F*, around both of which together it is wound three times to secure a good hold, and out to a a fleet sheave *C*, then back under another guide sheave, up over another head sheave, and down to the other cage *B*. This shows the arrangement for a two-compartment shaft. When the system is to be used for a single-compartment shaft, one end of the rope carries the cage and the other end carries a weight to balance the cage, and is run up and down

in a corner of the shaft. A balance rope is shown in the cut, and is generally used, though it is not essential to the working of the system, as it is in the Kœpe system. When sinking a shaft, a balance rope can not be used, as it is always in the way at the bottom of the shaft, where the work is being done.

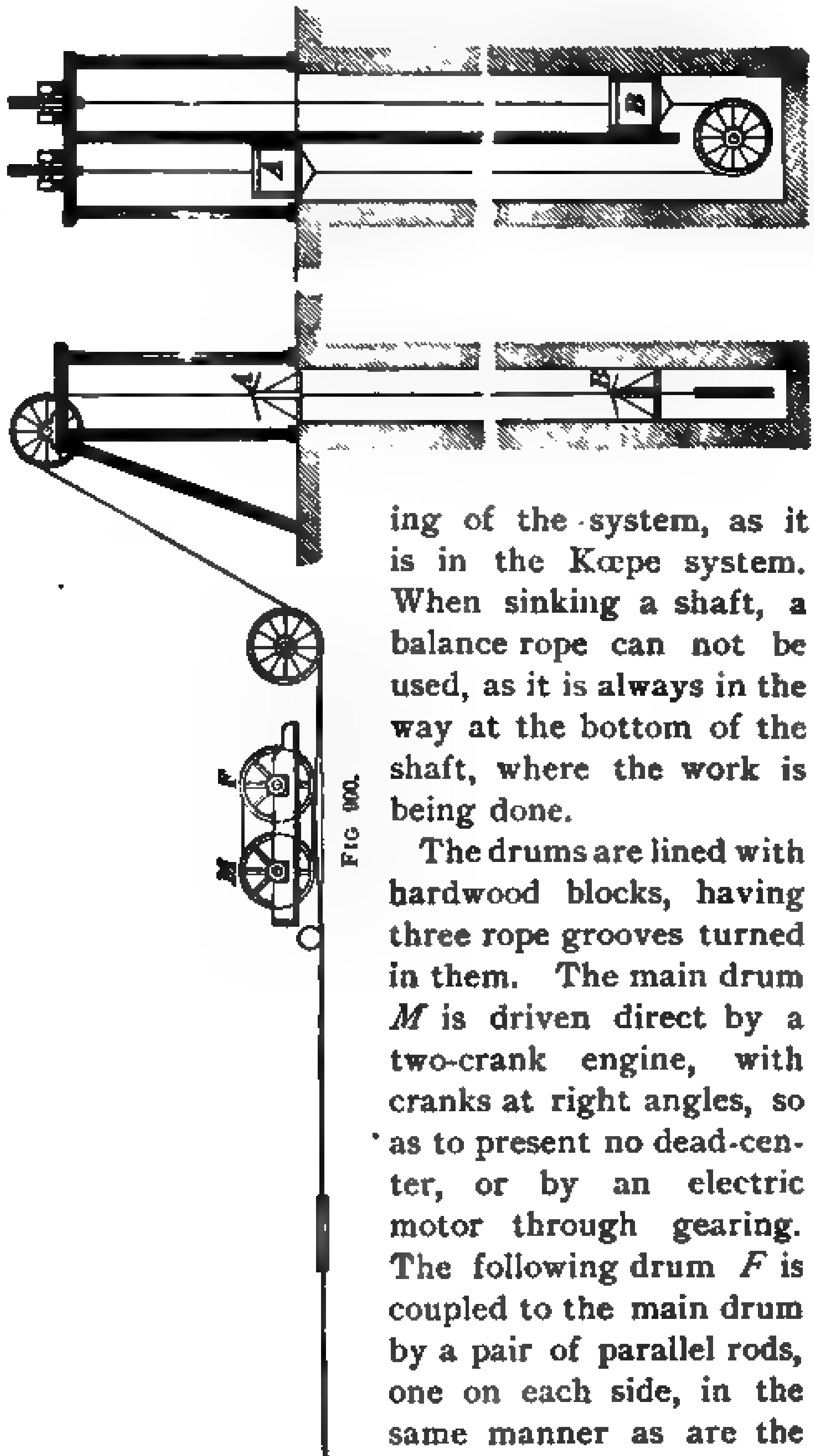


FIG. 900.

The drums are lined with hardwood blocks, having three rope grooves turned in them. The main drum M is driven direct by a two-crank engine, with cranks at right angles, so as to present no dead-center, or by an electric motor through gearing. The following drum F is coupled to the main drum by a pair of parallel rods, one on each side, in the same manner as are the drivers of a locomotive.

This gives six semi-circumferences of driving contact with the rope, as compared with the one semi-circumference of the Kœpe system. In the best plants on this system, the following drum *F* is tilted or inclined from the vertical an amount equal, in the diameter of the drum, to the pitch of the rope on the drum, the object being to enable the rope to run straight from each drum to the other without any chafing between the ropes and the sides of the grooves, and to eliminate the danger of the rope running off. This arrangement throws the shaft and crank-pins out of parallel with those of the main drum, but this is easily accommodated in the ends of the parallel rods.

The fleet sheave *C* is arranged to travel backwards and forwards, as shown in dotted lines, in order to change the working length of the rope. This makes the system very complete, and is desirable for the following reasons:

In sinking, as a greater depth is reached, it is necessary to let out rope in some way, that is, to increase its working length; in this system, it is done by moving in the fleet sheave towards the drums. If the shaft has been sunk 6 feet deeper, it is only necessary to move the fleet sheave in 3 feet nearer to the drums. Thus, it will also be seen that, if we are going to sink 500 feet during the life of the present rope, we will want a travel of the fleet sheave of 250 feet.

2499. In sinking and in regular hoisting, it is advisable to occasionally renew the fastenings at the ends of the rope, and to cut off a few feet of the ends where the greatest wear occurs. This shortens the rope, and the fleet sheave is moved in to make up the deficiency. A new rope stretches very considerably, and all ropes expand with the heat and contract with the cold. The length is therefore changing continually, and adjustment must be made in order to bring the two cages to their proper landings at the same time. This adjustment is easily made by moving the fleet sheave out or in.

2500. As was noted when considering the Kœpe system, in regular hoisting it is often necessary to hoist from

rope, the car, the cage, and the material to be hoisted applied at its circumference. In other words, if we have a hoist, as shown in Fig. 901, in which the diameter of the drum is 8 feet, the diameter of the rope is $1\frac{1}{4}$ inches, the length of the hoist is 1,500 feet, the weight of the car and the cage is 5,000 pounds, and the weight of the material to be hoisted is 5 tons, we will have a turning moment on the drum of 75,000 ft.-lb.

FIG. 901.

This result is obtained thus: Iron or steel rope $1\frac{1}{4}$ inches in diameter weighs 2.5 pounds per foot, or $2.5 \times 1,500 =$ 3,750 pounds for 1,500 feet. The total load would, therefore, be,

	3,750	pounds of rope;
	5,000	pounds of car and cage;
	10,000	pounds of material;
Total,	18,750	pounds.

This load would be applied at the circumference of the drum, or at a radius of 4 feet. The turning moment would then be $18,750 \times 4$, or 75,000 foot-pounds, and the direct pull on the brake, if it took hold at the same diameter, would be 18,750 pounds.

2506. If the hoist is double, and a tail-rope is used, the rope, car, and cage are balanced, and the brake will not have to be so powerful. Such a case is represented in Fig 902. We will suppose in this case that the same sizes of

drum, rope, car, and cage are used; that the shaft is of the same depth, and that the same weight of material is

to be h
the pre
be seen
the tw
balance eacn otner, as uo also the two cars and cages, and that the only load tending to turn the drum is that of the material. As before, this is 10,000 pounds, and the turning moment at the drum is $10{,}000 \times 4 = 40{,}000$ foot-pounds, as compared

FIG. 902.

with 75,000 foot-pounds, which we had before. The direct pull on the brake would be 10,000 pounds, as compared with 18,750 pounds.

This shows that the brake must be built to suit the case.

2507. There are two styles of brakes in general use; namely, **block brakes** and **strap brakes,** both of which are friction-brakes. A block brake is one in which one or more blocks, or shoes, being secured against a circumferential motion, are forced radially upon the drum surface and hold it against rotation by the friction between them. Such a brake is illustrated in Fig. 903, in which A is the drum, B

rope, the car, the cage, and the material to be hoisted applied at its circumference. In other words, if we have a hoist, as shown in Fig. 901, in which the diameter of the drum is 8 feet, the diameter of the rope is 1¼ inches, the length of the hoist is 1,500 feet, the weight of the car and the cage is 5,000 pounds, and the weight of the material to be hoisted is 5 tons, we will have a turning moment on the drum of 75,000 ft.-lb.

This result is obtained thus: Iron or steel rope 1¼ inches in diameter weighs 2.5 pounds per foot, or $2.5 \times 1,500 =$ 3,750 pounds for 1,500 feet. The total load would, therefore, be,

FIG. 901.

	3,750	pounds of rope;
	5,000	pounds of car and cage;
	10,000	pounds of material;
Total,	18,750	pounds.

This load would be applied at the circumference of the drum, or at a radius of 4 feet. The turning moment would then be $18,750 \times 4$, or 75,000 foot-pounds, and the direct pull on the brake, if it took hold at the same diameter, would be 18,750 pounds.

2506. If the hoist is double, and a tail-rope is used, the rope, car, and cage are balanced, and the brake will not have to be so powerful. Such a case is represented in Fig 902. We will suppose in this case that the same sizes of

drum, rope, car, and cage are used; that the shaft is of the same depth, and that the same weight of material is

to be h
the pre
be seen
the tw
balance each other, as do also the two cars and cages, and that the only load tending to turn the drum is that of the material. As before, this is 10,000 pounds, and the turning moment at the drum is $10{,}000 \times 4 = 40{,}000$ foot-pounds, as compared

Fig. 902.

with 75,000 foot-pounds, which we had before. The direct pull on the brake would be 10,000 pounds, as compared with 18,750 pounds.

This shows that the brake must be built to suit the case.

2507. There are two styles of brakes in general use; namely, **block brakes** and **strap brakes**, both of which are friction-brakes. A block brake is one in which one or more blocks, or shoes, being secured against a circumferential motion, are forced radially upon the drum surface and hold it against rotation by the friction between them. Such a brake is illustrated in Fig. 903, in which *A* is the drum, *B*

is the block, or shoe, being a piece of hardwood about two feet long; *C* is a brake-lever with fulcrum at *D*; *E* is a rod to transfer the leverage up to the engine-house floor where the engineer stands; *F* is an adjusting screw to take up the wear of the brake-block; *G* is the hand-lever by which the engineer operates the brake. One advantage in this kind of brake is that it requires very little motion of the shoe to free

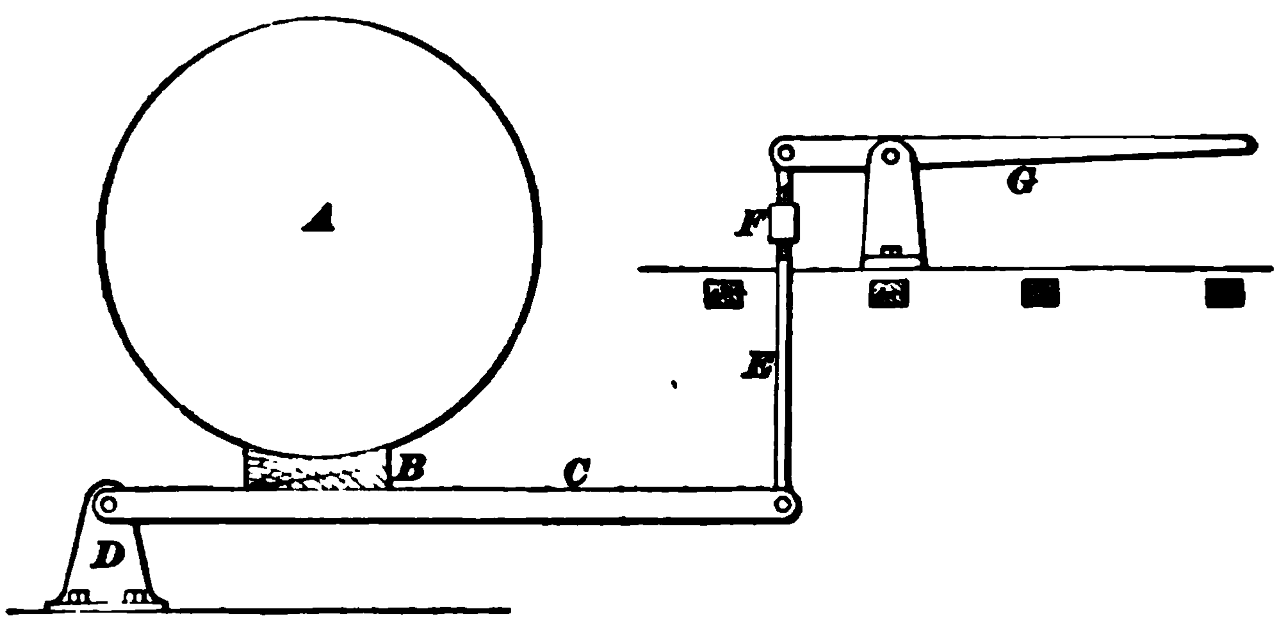


FIG. 903.

itself from the drum when the brake is off. This being the case, and the motion available at the other end of the brake gear being limited, it follows that the ratio of gearing can be great, and, consequently, also the force at the shoe.

This style of brake is very simple, but it is fast going out of use, because the single block gives so small an amount of rubbing surface, and the pressure of the brake, being entirely on one side of the drum, either lifts the drum or pushes it hard against its bearings.

2508. Block brakes with two blocks, commonly known as *post brakes*, give a greater amount of rubbing surface, and the pressure of the brake, being applied at two points diametrically opposite each other, is equalized. Such brakes are largely used in some localities. Fig. 904 illustrates a block brake. *A* is the drum; *B*, *B* are the blocks or shoes, which can be made long enough to encircle 90° on each side of the drum; *C*, *C* are the posts; *D*, *D* are the fulcrums, which should be tied together by a distance-piece, as shown, and should be securely fastened to the foundations;

E is a tension-rod, and *F* is a bent lever. Power is applied at the end of the lever *F*, as shown by the arrow.

An objection to this sort of brake arises from the fact that if the drum surface upon which the brake is applied is not

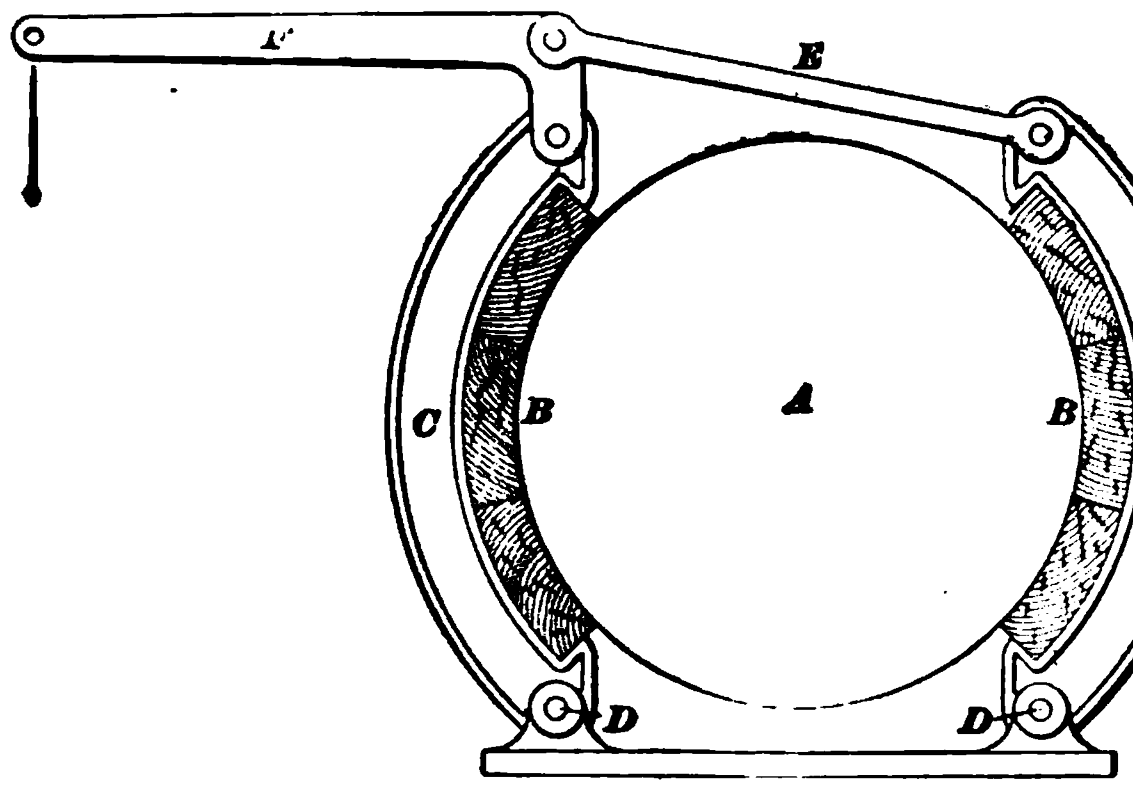


FIG. 904.

perfectly round, the resistance of the brake will not be uniform when applied while the drum is in motion.

Another objection to a block brake is that the wear is very excessive, owing to the small amount of surface in contact with the drum; and still another objection is the enormous force required to do a given amount of braking.

2509. There is a law of mechanics which states that the amount of friction between two surfaces varies directly as the pressure by which they are held in contact, and is independent of the area of those surfaces up to the point of abrasion. This law has been applied to the brakes of hoisting machinery, especially by those who advocate the use of block brakes; but it is not applicable, and for two reasons. In the first place, the operation of brakes is almost always at a pressure considerably beyond the point of abrasion, otherwise there would not be such wear both of the shoe and of the drum. In the second place, even inside of the point of abrasion, the law does not allow of a comparison between block brakes and strap brakes, because their surfaces are not similar. The surface of the block brake is short, rigid, and

practically flat, while the surface of the strap brake is long, pliable, and of a cylindrical shape. In the case of the strap brake, another law affects the results, a law which is familiar to us all in its practical application, and which we have all used when we have taken a couple of turns around a post with a rope, so that we could more easily resist some force at the other end.

2510. Strap brakes, in which the strap consists of a wrought-iron band, are used very extensively. The levers for transmitting the power from the hand-lever or treadle to the brake strap are variously arranged. In some cases the force is multiplied by several short levers, in others by one long lever. The treadle, however, has been replaced

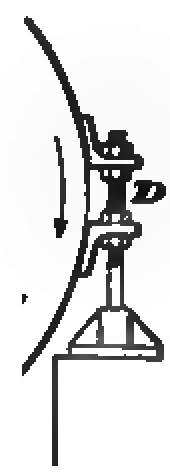


FIG. 905.

almost entirely by the hand-lever, and is now seldom seen except at old collieries. A single strap entirely surrounding the drum is sometimes used, but this is only satisfactory with small drums. The usual method is to have two straps, each extending half way around the drum. A brake of this sort is shown in Fig. 905, in which A is the upper brake strap and B the lower. One end of each strap is brought down to a bolt, one single and the other double, by riveting to the strap a forging a and a' as shown in the figure. The object in giving one end one bolt and the other two is to allow them to pass each other and yet have their lines of action intersect. These bolts are passed through a casting

C which is securely bolted to the foundation, and are fastened to it by four nuts on each bolt, two principal nuts and two jam nuts. This gives a means of adjusting the length of the strap to take up the wear. A second method of securing the back ends of the straps is shown at *D*. In this case a wrought-iron angular piece is riveted to each strap, and these are passed over the bolt that takes the place of the casting of the former arrangement. Nuts are used as shown, to adjust the straps for wear. The bolt should be short and stiff, so as to be well able to withstand the force tending to bend it when the drum is moving or tending to move in the direction shown by the arrow. The front ends of the straps are worked into eyes of somewhat less length than the width of the strap, as shown at *E*. These ends of the straps are fastened to the brake-lever *F* by means of bolts passing through their eyes and the brake-lever. The figure clearly shows the action of the brake. The brake-lever is supported on a pin at *G*, so that it can rotate about it. When the braking force is applied at *H*, through the connection shown, and in the direction of the arrow, the brake-lever tends to rotate, pulling down on strap *A* and up on strap *B*. These being held at the back end grasp the drum tighter as the force is applied.

2511. It will be noticed in Fig. 905 that the ends of the straps, both front and back, are brought in as close to

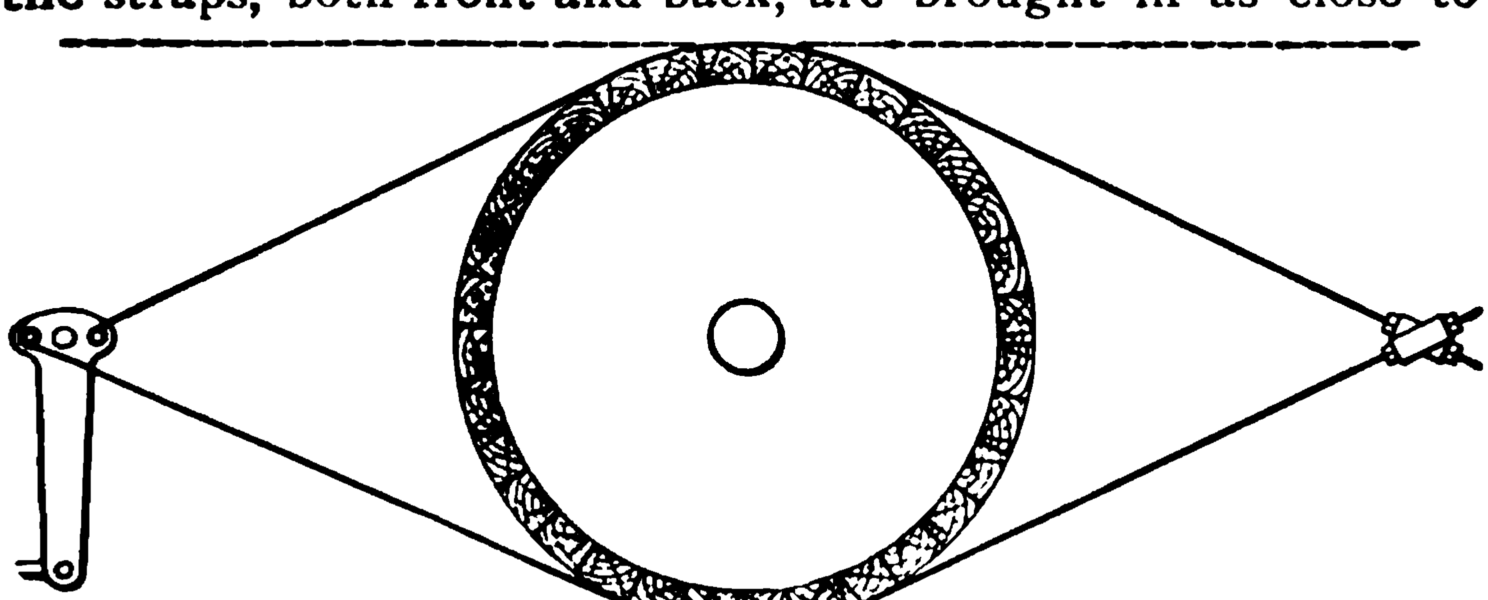


FIG. 906.

the drum as is practicable. This is done to give the greatest amount of contact between the drum and the straps,

and also to get the best effect from the force applied. Suppose, for instance, the brake were designed as shown in Fig. 906. It will be seen that here there is much less of the straps in contact with the drums than there was before, and also that the same pull on the straps will give much less pressure between the straps and the drum. This last idea may be more clearly understood if we imagine the straps to be lengthened indefinitely. Then we should have them practically parallel, as shown in dotted lines, and however great the pull, we should have no pressure between them and the drum.

2512. When brakes, such as we have been considering, are used on very large drums, they are found to be not stiff enough to lift themselves from the drum when the brake is off; for that purpose, springs are attached to the upper strap, so as to carry its weight. This idea is illustrated in Fig. 907, in which the springs at *A*, *A*, *A* are shown carried from the truss timbers by light wrought-iron rods.

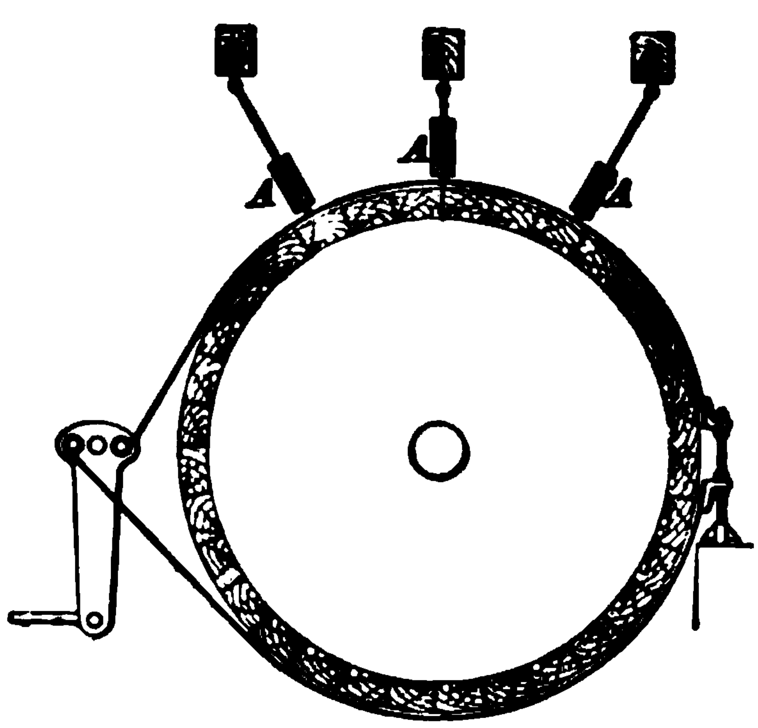


FIG. 907.

2513. So far, nothing has been said about the source of power to be applied to the brake. In the great majority of cases, it is hand power; but there are also many drums so large and winding such heavy loads that the power a man can exert by hand is not sufficient to brake them. Of course, the force that a man can exert can be multiplied indefinitely by levers and combinations of levers; but it must be borne in mind that, while the force is multiplied, the distance through which it can act is divided at the same rate. Now, it has been seen that a certain amount of motion is required at the brake band, in order to free it from the drum when it is off. This then, limits the leverage that a man can

use. Suppose, for instance, it is assumed that, with a strap brake, the band must come off from the drum half an inch. It would, therefore, have to increase in diameter 1 inch, or, say, 3 inches in circumference. Then, supposing a man to exert his force to advantage through 3 feet, or 36 inches, the leverage is $36 \div 3 = 12$. That is, if a man can pull 50 lb. on his hand-lever, he can exert 50 pounds $\times$ 12 = 600 pounds circumferentially on the brake band. This is with simple levers. If such a form of lever be adopted as will give a constantly increasing leverage, the force applied to the hand-lever will be multiplied in an increasing ratio. A diagram will explain this more clearly.

2514. In Fig. 908, *A* is the hand-lever, with a fulcrum at *B* and a pin at *C*, by which it takes hold of a reach rod or

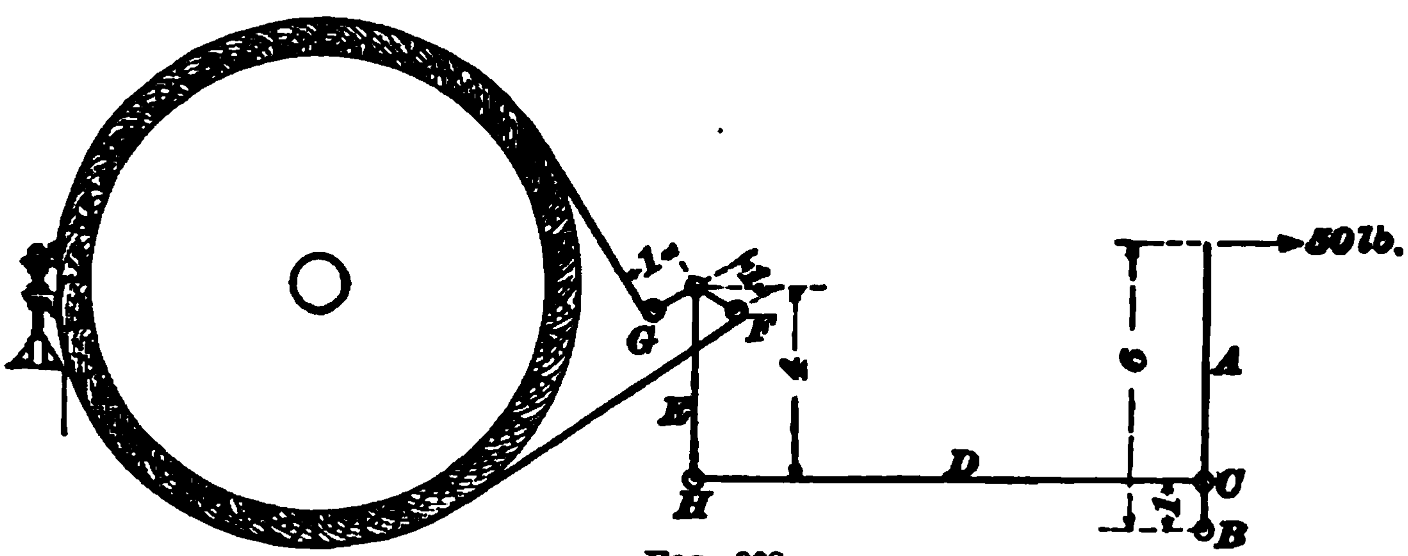


FIG. 908.

connection *D*. The connection extends to the brake-lever *E*, which has pins at *F* and *G*, by which the brake bands are operated. If we make the leverage of the hand-lever 6 to 1, and apply 50 pounds at its end, we will exert 300 pounds at the pin *C*, and, consequently, along the connection *D* to the end of the brake-lever *E*. Then, if we make the brake-lever with a ratio of 4 to 1, we will exert 300 pounds $\times$ 4 = 1,200 pounds at the pin *F* or *G*. This must be divided equally between them, however, which will give 600 pounds pull on each. Now, the distances through which these forces act are exactly in the inverse ratio of the powers. We have said that the brakeman can exert the force of 50 pounds through 36 inches. This, then, is the motion of the end of the hand-lever *A*. One-sixth of this, or 6 inches, will be the motion

at C, and therefore at H; one-fourth of this, or $1\frac{1}{2}$ inches, will be the motion at F and G. That is, F will increase its half of the brake band $1\frac{1}{2}$ inches in circumference, and G will do likewise with its half, making the total circumference 3 inches more, or the diameter 1 inch more, and thereby moving the band away from the drum $\frac{1}{2}$ inch radially. The levers are all shown in mid-position to make the figure more simple, but the relative leverages remain the same at all points in the motion.

This is a case of simple levers. Let us modify the scheme so as to have an increasing leverage, which is very easily done; but let us first realize that the *power* exerted at the hand-lever is just the same as that exerted at the brake band; that is, 50 lb. × 36 in. = 2(600 lb. × $1\frac{1}{2}$ in.); and no system of levers or other mechanism can make this any more or less. What we can do, however, is to make a system of levers that will give us a changing ratio between the forces at the hand-lever and at the brake band.

Fig. 909.

2515. In Fig. 909 we have shown our hand-lever to a larger scale. As before, B is the fulcrum, C is the pin where the reach rod takes hold, and the leverage is 6 to 1. The dotted lines show the extreme positions for a movement of 36 inches of the hand. Let us now take a length, as $B\,K$, for a radius to describe an arc of a circle about B. Then, from some point, as L, drop a perpendicular line and, parallel to it, another

at a distance from it equal to the horizontal motion of the point C. This second line crosses the circle at M. If $L M$ is greater than $P N$, we have taken L too far to the right. By trial, we can soon find a point L that is correct. Now, we have obtained the points for our new lever. Conceive the lever to be in its forward position and our reach-rod pin at L, instead of at C. We will then have a lever as shown at R in its forward position, and at S in its backward position. In the forward position, when only the clearance of the brake band must be taken up, and, therefore, but a small force required, the leverage is as 6 to $1\frac{1}{3}$, or $4\frac{1}{2}$ to 1, and in the back position, when the brake is on, and all the force is required that can be had, the leverage is as 6 to $\frac{1}{2}$, or 12 to 1. In other words, with this new hand-lever, a man, by exerting a pull of 50 lb. on the hand-lever, will pull with a force of 1,200 lb. on each of the brake bands, when the lever is in its back position.

2516. The question now arises, what will this amount do in the way of holding the drum against any force that may tend to rotate it? Such a force will resolve itself into a turning moment, and will be measurable in foot-pounds, as we have seen in the early part of this section. We must, therefore, transform the force that we get from the brake gear into another turning moment of the opposite direction.

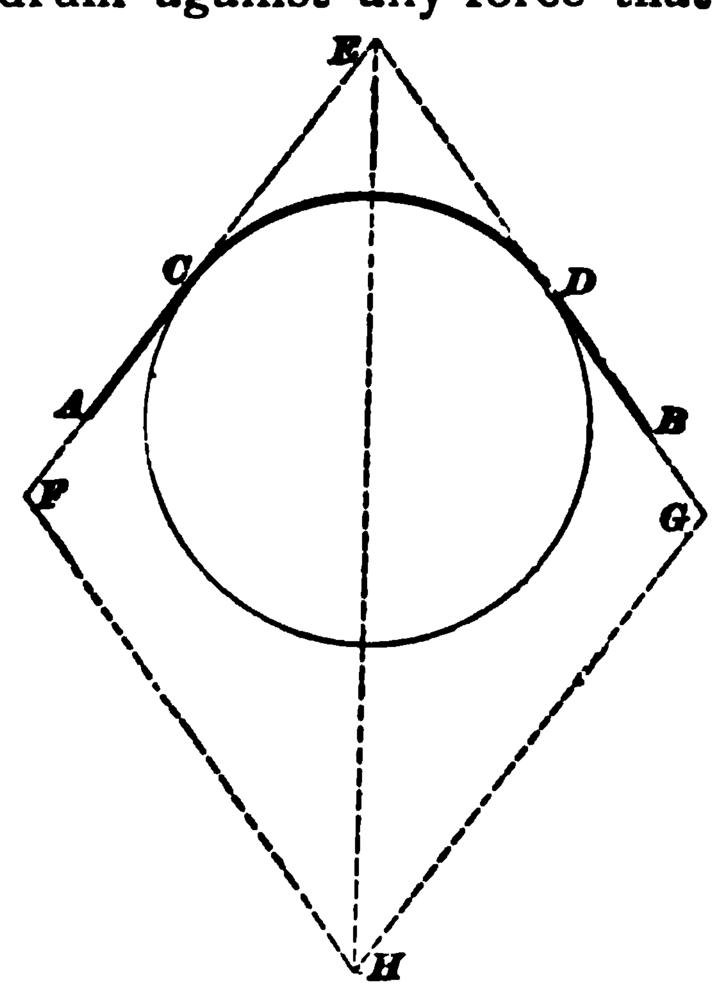


FIG. 910.

In Fig. 910 we have a drum and one brake band extending from a fixed end at A to a movable end at B, where the brake gear takes hold. These ends are some distance from the drum, so as to accommodate the anchorage at A and the brake-lever at B. We have, therefore, two tangents, A C

and $B D$. Extend these by dotted lines until they intersect at E, and from E lay off $E F$ and $E G$, proportional in length to the pull on the brake band. This we found to be 1,200 lb. acting at B, and we have an equal and opposite force acting at A. Using a scale of $\frac{1}{8}$ in. = 100 lb., and laying off these forces from E, we have $E F$ and $E G$ each $1\frac{1}{2}$ in. long. Completing the parallelogram of forces by drawing $F H$ parallel to $E G$, and $G H$ parallel to $E F$, we have the resultant $E H$, which is $2\frac{7}{16}$ in. long, and which, therefore, represents 1,950 lb. This is the pressure by which the brake band is held down on the drum. Now, we can count on about 40 per cent. coefficient of friction between a wrought-iron band and oak lagging, so our brake band would hold about $1,950 \times .40 = 780$ lb. The lower brake band would hold a like amount, making 1,560 lb. altogether. If this were applied to a drum 8 ft. in diameter, it would give a turning moment of 6,240 ft.-lb., or rather the power to resist that turning moment.

This is not the limit of the braking power of a man; but, from this we can readily see that there are many drums working under such loads that a man could not control them by hand. We must, then, resort to steam, compressed air, or water under pressure. At first, this seems to be very simple. Referring to the brake gear of Fig. 908, it will be remembered that we had a force of 300 lb. along the reach rod D to the brake-lever. Later, when we used the hand-lever of Fig. 909, we had 600 lb. along this connection, but we saw that even this would not be enough in many cases. Suppose that we wished a force of 6,000 lb. instead of 600 lb. If we attach this connection to a piston-rod of a small engine, we can easily get whatever force we may wish. Let us assume that we are carrying 100 lb. of steam in our boilers. This we can pipe to the cylinder of our small engine, and, by means of a valve, admit it to the cylinder whenever we want it. To get a pressure of 6,000 lb. from steam at 100 lb. pressure, we would require $6,000 \div 100 = 60$ square inches of piston area. Such a piston would be $8\frac{3}{4}$ inches in diameter. Our arrangement would then be as shown in Fig. 911. The hand-lever A, through its connec-

tion, opens the steam-valve v on the cylinder. This admits steam to the front end a, which drives the piston back, and so puts on the brake. The mechanism and its operation would be practically the same for either steam, compressed air, or water under pressure; and so far the arrangement is satisfactory. But when we come to use such a brake as this, we find that the action of steam or any other medium under such a pressure is so rapid that the brake is applied with its full force almost instantaneously. This subjects the various parts of the plant to very severe strains, which are objectionable. In fact, it is not permissible. One scheme to modify this action which has been used successfully is the use of the steam-valve that requires a long travel to give it a full opening. Such a valve can be opened a little, so as to allow the steam to leak through, and thereby

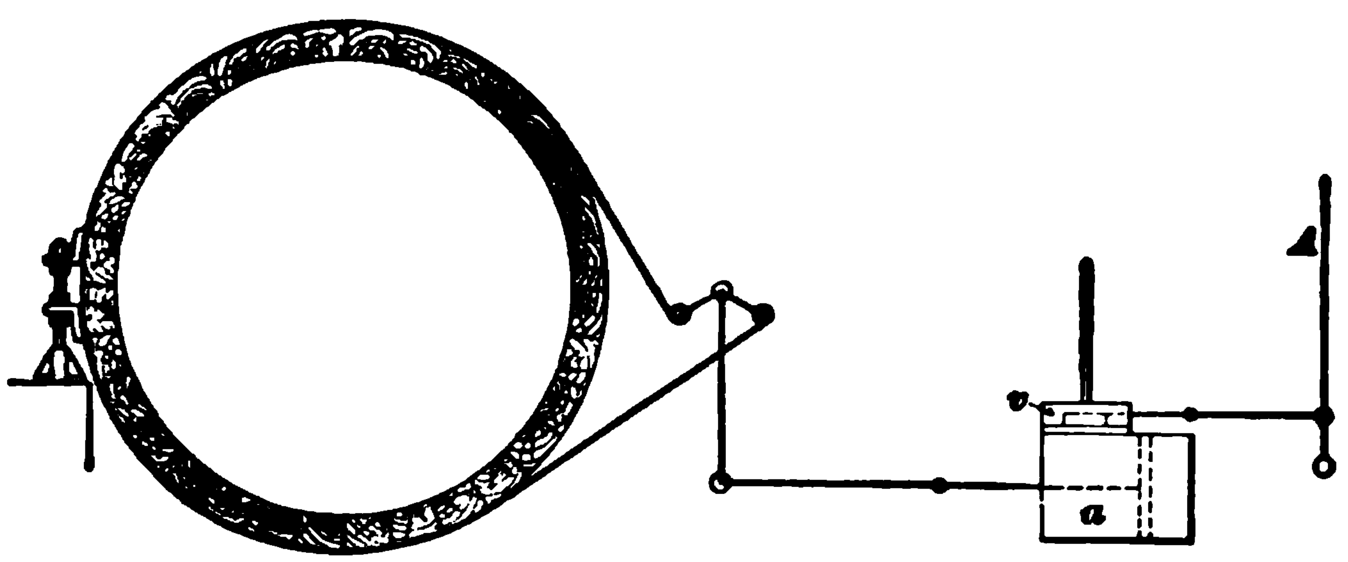


FIG. 911.

increase the pressure in the cylinder gradually. If such a valve, for the inlet of steam, is coupled to another working in the opposite direction for the outlet of steam, the operation is very successful. With this arrangement, as the inlet valve begins to open, the outlet valve begins to close. At first, this will simply pass a little steam through the cylinder without allowing any pressure to accumulate there. As the valves move farther, the amount of steam admitted increases, while the size of the outlet decreases, and as a consequence a pressure forms in the cylinder. This action continues until, when the inlet valve is full open and the outlet valve is entirely shut, the full pressure of steam is in the cylinder and the brake is full on. By reversing the motion of the valves the brake comes off.

CLUTCHES.

2517. In most hoisting plants there is no need of a clutch, because the motor is geared directly to the drum and operates only when it is necessary to make a hoist. Sometimes, however, several drums are operated by one motor or engine; it then becomes necessary to have a clutch fastened to the motor, by which it can take hold of the drum at any time. This principle of operation has been adopted in some very extensive hoisting plants in the copper regions of Lake Superior. The engine runs continuously and drives

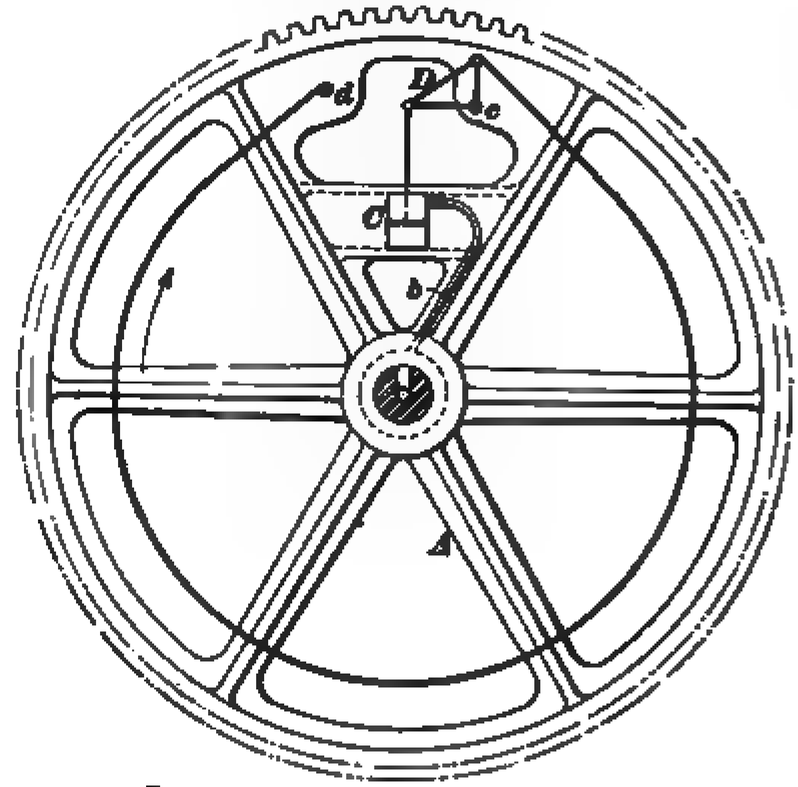


FIG. 912.

a clutch wheel on each drum. These clutch wheels are virtually strap brakes mounted on revolving wheels instead of on a steady foundation, as usual, and when they take hold of their drums they carry them around with them. The principle of these is illustrated in Fig. 912. There is a spur-wheel A, which is driven continuously by the engine through a spur pinion, not shown in the figure. This spur-wheel is mounted on a shaft which carries the gear around with it.

The shaft rests in two bearings, one next to the wheel, as shown at a, and the other at the other side of the drum.

Part of the drum is shown in the left-hand view, but is removed in the right-hand view in order to show the clutch more clearly. In the drum shaft there is a hole extending from one end through the center to the middle of the clutch wheel, and then out to the surface, as shown at *B*. From the end of the shaft this hole is connected by piping to a hydraulic valve operated by the *brakeman*, so that he can at will admit water or oil under pressure into it. The water comes out of the shaft into a recess in the hub of the clutch wheel, and from there is piped by *b* to the upper end of a small cylinder shown at *C*. Here it forces the piston down, or rather towards the center of the wheel, for it must, of course, operate irrespective of the position of the wheel; the piston carries with it one end of the bell-crank *D*. To the other end of this bell-crank is attached one end of the clutch band, the center *c* of the crank being carried by a pin secured to the clutch wheel. The other end *d* of the clutch-band is also held by a pin from the clutch wheel, and through this pin the drum is driven. As the piston moves towards the center of the wheel, it rotates the bell-crank, and through it pulls on the end of the clutch band. The other end being secured, this decreases the diameter of the band and makes it grip the drum, which is just inside of it, and which must, therefore, go around with it. The direction of motion is shown by the arrow. When the hoist has been made, the brakeman, by closing his valve, removes the pressure in the system and the clutch band comes off, being assisted by springs. The drum must then be controlled by a regular brake taking hold of its other end. One peculiarity of a drum built for this purpose is that it has a continuous hub throughout its entire length. This is babbitted like a journal-box or bearing, and is supplied with an oiling device, because it revolves on the shaft.

Of course, when a hoist is being made, the drum is carried around with the clutch wheel and shaft, and there is no motion between it and the shaft; but after the hoist has been made and the drum is standing still or is running the other way to lower the cage, there is motion between them,

for the wheel and shaft continue in the same direction as before In a hydraulic clutch there should be some elastic body between the force that the brakeman applies and its point of application. A simple method is to put a spring between the end of the clutch band and the bell-crank to which the clutch band is attached. The object is to prevent the entire force of the grip taking effect at once.

When the clutch is put on, there is slipping between it and the drum until the latter has acquired the same velocity as the former. This gives time in which to overcome the inertia of the moving parts. If the drum is started too suddenly, the strain on the rope becomes so enormous that it is liable to break.

HOIST INDICATORS.

2518. A hoist indicator is a piece of mechanism forming part of the drum gear, designed to show the engineer the position of the cage or cages at every point during the hoist. In some form or other, these are quite generally used, though not universally. In some cases they are essential, while in others they can be omitted. The objection to their use is their liability to get out of order. Almost every piece of mechanism is liable to get out of order, and if an engineer is relying on his hoist indicator to make a landing, and it fails him, there is likely to be overwinding, with possibly very serious results. If it is possible, there should always be some additional way of locating the landing by which the engineer can check the readings of his indicator, and thus avoid a possible accident. In some cases, this second method may be better than the indicator and sufficient in itself.

2519. A very common and simple hoist indicator is made by inserting a pin of suitable diameter into the center of the end of the drum shaft, and using this as a miniature drum upon which to wind and unwind a chain or cord corresponding to the hoisting rope. This chain or cord is then led over a sheave or pulley at the top of a pair of guides

representing the hoist, and carries at its end a weight, pointer, or gong, which serves as the car or cage. The landings are then marked on the guides, and the whole is placed immediately in front of the engineer. If a gong is used, which is preferable, a pointer may also be added; and the gong is so arranged that it will be struck at a point some distance short of the landing, to call the engineer's attention.

This style of hoist indicator can not, however, be called good. It is in use in many places, partly because it is cheap, and partly because the men who do the hoisting depend upon a mark on the rope and not upon the indicator.

If indicators are used, they should be trustworthy and not liable to derangement from trivial causes. In other words, they should have a positive motion, driven by gearing from the hoisting machinery. For the exact location of the cage at the top of the shaft, most engineers rely upon the white marks made on the last coil or coils of the rope.

This plan he considers more accurate, more trustworthy, and less likely to lead him into error than any indicator which can be constructed. This faith in the rope as an indicator has spread to many mining engineers and superintendents, so that we now find many large collieries not provided with indicators. Although this use of the rope is good and should be adopted wherever possible, as a second indicator, to check the regular one, we do not recommend its use to the exclusion of an indicator.

2520. A very good indicator with a positive motion, and simple in its construction, is shown in Fig. 913. It consists of a worm A secured to the drum shaft B, engaging with and driving a worm-wheel C. This wheel is mounted on a shaft D, which is supported in the bearings E and E, and carries at its end a pointer F. There is a dial-plate G slipped over the end of the shaft and screwed to the forward bearing. This is just behind the pointer, and the different levels are marked on it, around its circumference, so that when the pointer indicates any one of them it will mean that the cage is at that level.

Let us suppose that we want an indicator for a shaft 800 ft. deep from rail to rail; that the drum to be used is 10 ft. in diameter, and that the drum shaft is 10 in. in diameter. The circumference of the drum is 31.42 ft.; hence, the revolutions per hoist are $800 \div 31.42 = 25.46$ revolutions. Then, if we let the pointer make one revolution per hoist, the ratio of our gearing will be 25.46 to 1. Let us, however, have it make a little less than one revolution per hoist, so that our velocity ratio will be as 25 to 1; then we can use a single-threaded worm, and our wheel will have 25 teeth. If we make the pitch of the teeth $1\frac{1}{2}$ in., the circumference of the wheel will be $25 \times 1\frac{1}{2}$ in. $= 37\frac{1}{2}$ in., and

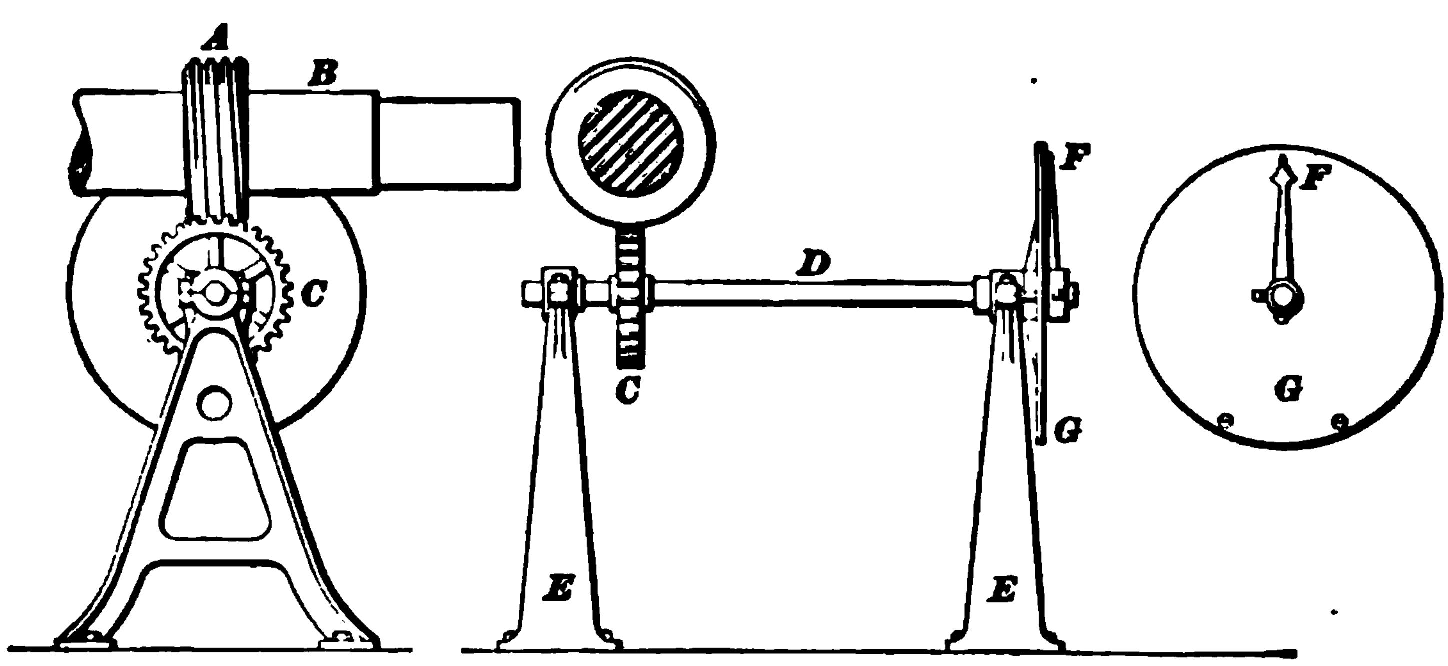


FIG. 913.

the diameter will be $11\frac{15}{16}$ in. The pitch of the worm screw will, of course, be the same as that of the wheel, and its diameter will be whatever is necessary to give sufficient strength outside of the shaft, since it bears no relation to the velocity ratio.

2521. One fault of nearly all indicators is that they give a regular movement throughout the winding, and the space over which the pointer travels is too small to enable the engineer to land the cage accurately. There are some exceptions to this, however, as indicators have been made with a differential motion to the pointer, which was greater at the time of landing and less during the middle of the hoist. They have also been made with two pointers, one to operate like the one in the hoist above described, and the

other to remain stationary during all of the hoist but the last few feet, then to start and move all the way around its circle during those few feet of the hoist. These have not been adopted to any extent, however, and reliance has mostly been placed on a mark on the rope to locate the landing accurately.

ROPE FASTENINGS.

2522. To fasten the rope to the drum, a very common practice is to pass the rope through a hole in the drum rim and clamp it to the drum shaft. Such a fastening is shown at *a* in Fig. 914, as applied to a wood-lagged drum. Care should be taken in such a case to make the radius of curvature of the hole at *A* as large as possible within the thickness of the lagging, so that the rope will not be bent any sharper than is necessary. When an iron drum is used, the thickness of the rim does not afford enough depth in which to bend the rope, and it is necessary to build in a pocket for the purpose, as shown at *b* in the figure.

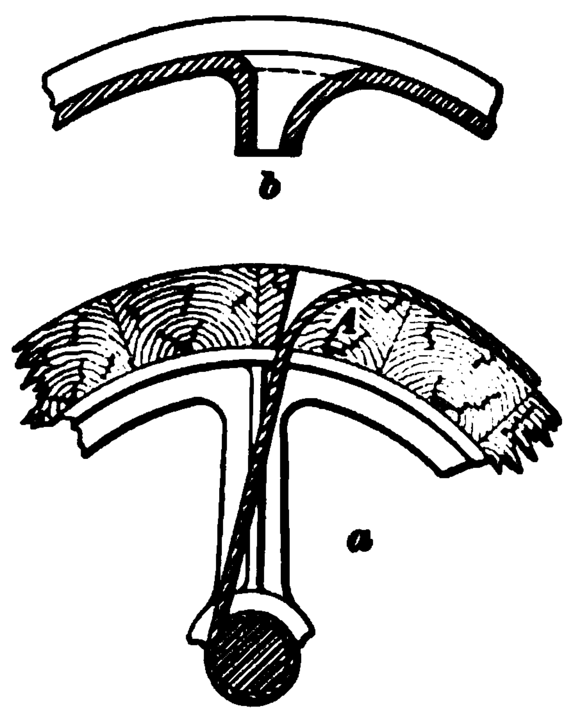


FIG. 914.

ROPES.

2523. Round wire ropes have already been considered in Mine Haulage, and as, in hoisting, flat and tapered wire ropes are also used, we will only consider these here. In the use of ropes we have also to consider certain details, as follows: *Rope-ends; Detaching Hooks; Tail-Ropes.*

2524. Iron or Steel Ropes.—As a choice between iron and steel for ropes, steel is the better. In the first place, a steel rope is lighter than an iron rope of the same strength, and, consequently, makes less weight to be handled during the hoisting. Weight here is objectionable because it is a part of the moving mass, and, therefore, requires power to put it into motion and to stop it again. It is also

objectionable because it requires stronger sheaves in the head-gear and drums in the engine-house and larger shafts and bearings wherever it is carried.

In the second place, a steel rope is smaller than an iron rope of the same strength, and, therefore, can pass around smaller drums and sheaves than the iron rope.

To illustrate these ideas, suppose we have a vertical shaft 1,800 ft. deep, and are required to lift at the end of the rope 10,000 lb. Now, a wrought-iron rope $1\frac{1}{2}$ in. in diameter weighs 3.65 lb. per foot; hence, 1,800 ft. of it would weigh $1,800 \times 3.65$, or 6,570 lb. This, of course, must be added to the load to be lifted to get the total load on the rope, which is, therefore, 10,000 lb. + 6,570 lb. = 16,570 lb., or, say, $8\frac{1}{4}$ tons. The breaking strength of a $1\frac{1}{2}$-inch wrought-iron rope, having 19 wires to a strand, is 39 tons, which gives a factor of safety of $39 \div 8\frac{1}{4} = 4.7$.

On the other hand, a cast-steel rope 1 in. in diameter, having 19 wires to the strand, weighs 1.58 lb. per foot, or 2,844 lb. per 1,800 ft., which would make the total load 12,844 lb., or, say, $6\frac{1}{2}$ tons. The breaking strength of a 1-inch cast-steel rope, having 19 wires to a strand, is 33 tons, and this gives a factor of safety of 5.1, nearly.

For this case, then, the wrought-iron rope weighs 131% more than the cast-steel rope and gives a lower factor of safety. Furthermore, the minimum diameter of the sheave or drum upon which it would be safe to work the wrought-iron rope is $1\frac{1}{2} \times 60 = 90$ in., or $7\frac{1}{2}$ ft.; whereas the cast-steel rope would work safely on a drum or sheave $1 \times 60 = 60$ in., or 5 ft. in diameter.

The cast-steel rope which we have used here is not the highest grade of rope. There are special high-grade ropes made of plow steel, which are considerably stronger, and a comparison between them and the wrought-iron ropes would show a still greater difference in favor of steel. A plow-steel rope $1\frac{1}{2}$ in. in diameter, the same size as the above wrought-iron rope, has a breaking strength, as will be seen by referring to Table 46, of 110 tons, almost three times as much as the wrought-iron rope.

2525. Round Ropes.—Round ropes are much more generally used than either flat or tapered ropes. There are two kinds of them manufactured, one being more pliable than the other. The first of these contains nineteen wires in a strand, and has six strands wound around a hemp center. These are generally used for hoisting and power transmission because they are very pliable. The other kind contains seven wires in a strand, and has six strands wound around a hemp center. These are not generally used for hoisting because they are not pliable. They may be used, however, if necessary, and if extraordinarily large drums and head sheaves are used. When ordering a rope, the use to which it is going to be put should be stated to the maker, as his advice on the subject is valuable. Experience has demonstrated that the wear of a rope increases with the speed at which it is worked. It is therefore advisable to increase the load rather than the speed, if increased capacity is desired.

2526. Flat Ropes.—Flat ropes are composed of a number of strands, alternately twisted to the right and to the left, laid alongside of each other, and sewed together with soft iron wires. They are wound upon the vertical drums, or reels, which have been previously described, because, after the rope has coiled once around the drum, it coils upon itself and piles up vertically instead of spreading out horizontally. Such ropes and drums are used at many mines in the western part of the United States, and at some mines in Europe, but they are practically unknown in the anthracite coal regions of Pennsylvania.

Many advantages have been claimed for them that have not been proved by actual service. The counterbalancing action due to the rope coiling on itself is an advantage, and if the diameter of the drum be made small enough a remarkable uniformity in the load may be obtained. When the rope is all out, and so presents the greatest resistance, due to its weight, it winds upon a small diameter, and its leverage is, therefore, small; when it is wound up and

presents the least resistance, it winds upon a large diameter and its leverage is large.

As was stated in studying drums, it is also an advantage to have a reel as compared with a cylindrical or conical drum, on account of the short drum shaft that can be used, and the consequent bringing together of the engines. A reel is also less costly to build.

All the advantages, however, that can be counted up in favor of flat ropes are more than counterbalanced by the excessive wear of the rope, due to its coiling upon itself, to the nature of its construction, and by its greater first cost.

2527. In the following table are given the sizes, weights, and breaking strengths of steel flat ropes:

TABLE 49.

FLAT STEEL ROPES.

Size. Inches.	Weight per Ft. Pounds.	Breaking Strength. Pounds.	Size. Inches.	Weight per Ft. Pounds.	Breaking Strength. Pounds.
$\frac{3}{8} \times 2$	1.27	38,500	$\frac{1}{2} \times 3$	2.57	76,500
$\frac{3}{8} \times 2\frac{1}{2}$	1.52	46,000	$\frac{1}{2} \times 3\frac{1}{2}$	3.00	89,500
$\frac{3}{8} \times 3$	2.02	61,000	$\frac{1}{2} \times 4$	3.43	102,000
$\frac{3}{8} \times 3\frac{1}{2}$	2.28	69,000	$\frac{1}{2} \times 4\frac{1}{2}$	3.86	115,000
$\frac{3}{8} \times 4$	2.53	76,500	$\frac{1}{2} \times 5$	4.29	127,500
$\frac{3}{8} \times 4\frac{1}{2}$	3.04	92,000	$\frac{1}{2} \times 6$	5.15	153,000
$\frac{3}{8} \times 5$	3.29	99,500	$\frac{1}{2} \times 7$	6.01	178,500
$\frac{3}{8} \times 6$	4.05	122,500	$\frac{1}{2} \times 8$	6.86	204,000
$\frac{5}{8} \times 4$	3.96	143,000	$\frac{3}{4} \times 5$	5.81	183,500
$\frac{5}{8} \times 4\frac{1}{2}$	4.62	166,500	$\frac{3}{4} \times 6$	7.74	245,000
$\frac{5}{8} \times 5$	5.28	190,500	$\frac{3}{4} \times 7$	8.71	275,500
$\frac{5}{8} \times 6$	6.60	238,000	$\frac{3}{4} \times 8$	9.68	306,000
$\frac{5}{8} \times 7$	7.26	262,000	$\frac{3}{4} \times 9$	11.62	367,000
$\frac{5}{8} \times 8$	8.58	309,500	$\frac{3}{4} \times 10$	12.58	398,000
$\frac{5}{8} \times 9$	9.24	333,000	$\frac{3}{4} \times 11$	13.55	428,500
$\frac{5}{8} \times 10$	10.56	381,000	$\frac{3}{4} \times 12$	15.49	489,500

2528. Taper Ropes.—A taper rope is one made tapering from end to end, with the idea of giving it uniform strength while carrying its own weight. It is less in diameter and less strong at the cage end, where the load consists only of the cage, car, and its contents, and it is greater in diameter and stronger at the drum end, where the load consists of the cage, car, its contents, and the weight of the rope itself. Ropes of this kind, both round and flat, have been used to some extent. The idea is correct, theoretically, and a rope made thus will be stronger for a given weight than an ordinary rope, but, practically, the scheme is not a good one. A taper rope to do a given amount of work costs more than a straight one, and owing to the difficulty of manufacture it can not be made so perfect. Besides, it will not work well over the sheaves and drums, because of its changing diameter. In fact, it is not a practical rope, and, therefore, it does not require our attention.

2529. Rope-Ends.—One of the vital points of a hoisting plant is the attachment to the end of the rope. The bucket, car, or cage has fixed to it two or more lengths of chain terminating in a ring or shackle, and it is necessary to fasten to the end of the rope something by which a secure though temporary connection can be made to this shackle. It is generally called a **capping**, and exists in many different forms. The old plan was to employ two semicircular collars encircling the rope, these being prevented from slipping or drawing off by rivets which passed both through the rope and the capping. The driving of these rivets necessarily injured the ropes, and to remove this objection, a capping consisting of two halves of a conical sleeve with collars driven over it, as shown in Fig. 915, was used. The two parts of the sleeve are one piece with the link at *A*. They do not come together when closed over the rope; so when the rings *B*, *B*, *B* are driven down over the cone the two

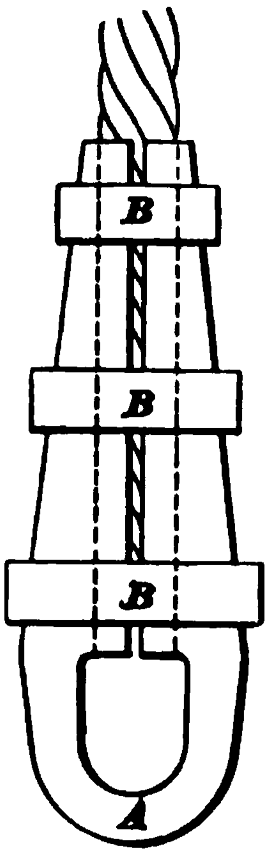


FIG 915.

parts of the sleeve clamp the rope. The objection, however, is that there is not a positive hold taken of the rope, but reliance is placed on the friction of the clamp.

The best rope-end in use is a wrought-iron or steel conical socket, as shown in Fig. 916. This is in one solid piece. To attach it to the rope, the rope is first threaded through from the small end and allowed to project a short distance.

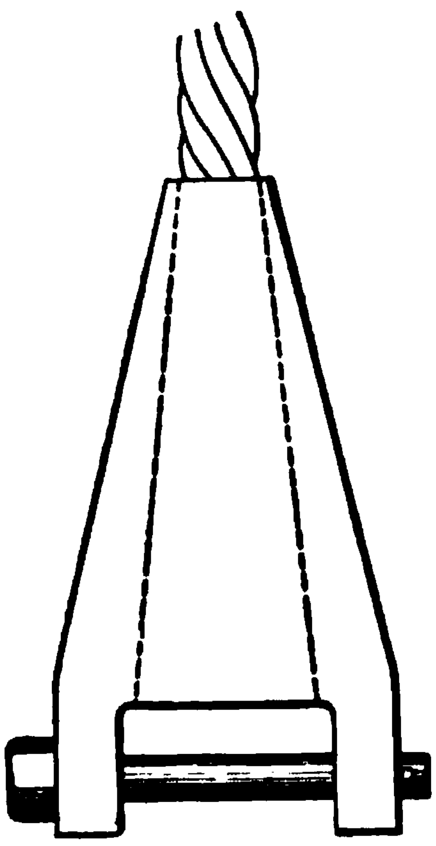


FIG. 916.

The ends of the strands are opened and bent back on themselves, part of each strand being cut away. This makes the end of the rope conical, and in that condition it is drawn back into the socket. As an additional security, a conical wedge is often inserted in the place originally occupied by the hemp core. Except under abnormal conditions, it is impossible to draw the thick end of the rope through the small end of the socket without first splitting it. If properly constructed of suitable material, such a thing could scarcely happen. For very heavy loads, collars are shrunk on. At the point where the rope leaves the capping, the wires are subjected to a sharp bending action, and often break. It is, therefore, necessary that careful inspection should often be made at this point. A plan is adopted at some collieries of re-capping the ropes at regular intervals, whether they appear to require it or not. In wet shafts the wires rust inside of the capping, and such action can not be detected. To prevent it, the capping is sometimes run full of lead, which is a very good scheme.

2530. Detaching Hooks.—In all hoisting, there is more or less danger of overwinding when the car is lifted too far, and it is then dashed more or less violently against some timber or other obstruction. Various schemes have been adopted to automatically prevent this, chief among which are the detaching hooks; although it can not be said that any of them have been a decided success. In the

United States, and especially in the eastern coal fields, there is a strong feeling against the adoption of any such device. It is held that they inspire the engineer with a misleading feeling of security; that they are more or less complicated in construction, and so need care, and destroy the simplicity of the plant; that they may be the direct cause of accident by introducing new elements of danger; that they add to the cost, and that they are not thoroughly reliable.

Again, it is held that the surest prevention of overwinding is obtained by the employment of a sober, reliable, and competent engineer, who is held personally responsible for overwinding accidents; by having a good brake and an engine thoroughly under the control of the engineer; by a reliable method of indicating the position of the cage, whether by hoist indicator, by mark on the rope, or by both, and by giving sufficient height to head sheaves to allow of considerable hoisting over and above that necessary for landing.

2531. In England, detaching hooks are used quite generally. They were originally of two kinds: those which simply detached the rope, and those which at the same time prevented the cage from falling; but, since additional means had to be provided in the former case for holding the cage, they have given way to the latter kind. There are several such hooks to be had, which differ from each other only in the smaller details. In all of them, detachment is effected by passing the rope through a circular hole in an iron plate, or through an iron cylinder, the size of which is sufficient to allow a portion of the hooks to pass through in its working state, but not to allow it to fall back again when disengagement has taken place. Such a hook is shown in Fig. 917.

It consists of two outside fixed plates A and A, enclosing between them two inner movable ones B and B, which can oscillate about a strong pin C passing through both plates and framework. The upper ends of plates A and A are made of uniform width, somewhat less than the diameter of the hole in the disengaging plate; but near the bottom there

are two projections D and D, to prevent the hook from passing entirely through the hole. The winding-rope is attached to the top shackle E, and the cage to the lower one F. When the two movable plates are placed on the central bolt, their upper parts close in opposite directions upon the pin of the shackle and entirely overlap it. In this position, they are secured by a copper pin G. In case of overwinding,

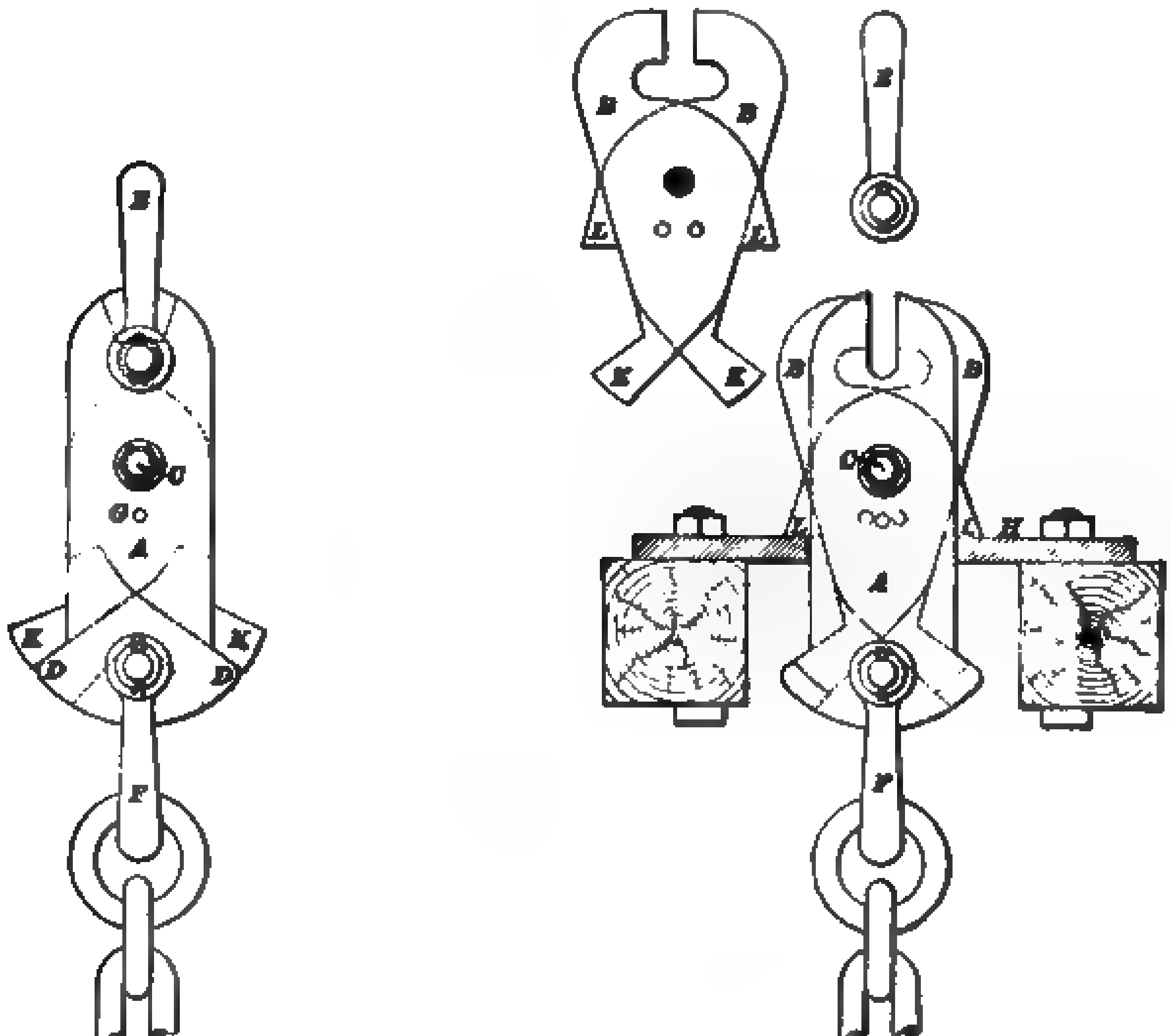


FIG. 917.

when the hook passes into the hole of the disengaging plate H, the two projections on plates B and B, shown at K and K, are forced inwards, turning the plates about the central bolt C. This shears the copper pin G and at the same time releases the shackle E. The inner plates are then in such a position that the projections on them at L and L can not pass down through the hole again. The cage then hangs by the hooks to the disengaging plate, and the rope goes on to the drum until it stops winding. An objection that can be

raised against this hook is that, being constructed of plates, there is considerable surface in contact between the moving parts, and unless they are regularly taken apart and oiled, there is danger of their rusting firmly together.

2532. Tail-Ropes.—When cylindrical drums are used for hoisting, perfect counterbalancing of the weight of the rope can be secured by several methods; but they have all

Fig. 918.

given way to the endless-rope system, for it has proved preferable to all others. It consists in attaching to the under side of one of the cages a tail-rope equal in size to the hoisting rope; and after running it down the shaft into the sump, where it forms a semicircle, returning it up, and attaching it to the under side of the other cage. Such an

arrangement is shown in Fig. 918. When this system was first introduced, it was thought that a pulley, or sheave, must always be placed in the sump for the tail-rope to pass around, the sheave being free to move up and down with any change of length of the rope, but secured against any other motion. The sheave shaft rotated in bearings arranged to move up and down in vertical guides, and the weight of the whole affair was carried on the rope. In many cases, however, no such pulley is used. All that has been done to keep the tail-rope from twisting is to fix two beams, side by side across the shaft in the sump, between which the tail-rope passes, and another one below this in the opposite direction passing through the loop in the rope. This arrangement is shown in Fig. 919. It is, perhaps, preferable to use a guide-pulley in the sump, as old winding-ropes can then be used. Otherwise, a special rope has to be employed, as old winding-ropes are not sufficiently flexible to run around such a loop without the assistance of a pulley. The tail-rope is connected to the bottom of the cage by any ordinary capping, with a bolt passing through it, and a cross-piece of the cage. By this system, perfect counterbalancing is obtained, as a factor is introduced which is equal and opposite to the hoisting rope. An objection to its use lies in the fact that a greater weight is put upon the capping of the hoisting rope, which is liable to be a weak part of the apparatus. Great care should, therefore, be taken to see that the capping is as strong or a little stronger than the rope itself. The load on the head sheaves is no more than the maximum load without the tail-rope; but it is a uniform load, and, therefore, possibly preferable.

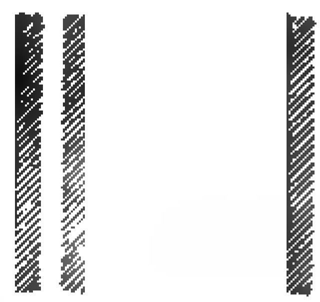


Fig. 919.

The power of the motor need not be so great when a tail-rope is used as when no tail-rope is used, because it does not have to lift the weight of the hoisting rope hanging in the shaft, this being balanced by the tail-rope; but, on the other

hand, the inertia of the moving parts is greater, due to the mass in the tail-rope.

Sometimes difficulties arise from the fact that the vertical center lines of the two cages are very near together, and that, therefore, the sheave that can be used in the sump to guide the tail-rope is of too small a diameter. To obviate this difficulty a chain is sometimes used instead of a rope.

CARS.

2533. We use the term car here in its broadest sense, covering whatever is on the end of the rope. In practice, however, the term used in that way would not be definite enough. In studying the different styles, we will do so under their specific names, that is to say: *Buckets; Cross-heads; Cars; Cages; Gunboats, or Skips.*

BUCKETS.

2534. The simplest sort of a car for hoisting material from a mine is what is generally called a **bucket.** It is used to a great extent, both in small operations permanently and in large operations for sinking purposes. Fig. 920 represents such a bucket. They are usually made of boiler-iron, about three feet in diameter at the top, two feet six inches in diameter at the bottom, and of about the same depth. They are suspended by a handle, pivoted slightly below the center of gravity, and are locked in an upright position by loose rings on the handle which slip over pins on the rim of the bucket, as shown in the figure. When the hoist has been made and it is desired to empty the bucket, this ring is lifted up so as to release the bucket at the top, when it will upset of itself. It is well to use two such buckets for regular service, so that one can be left at the bottom to be filled while the other is hoisted with its load. This saves

FIG. 920

the time occupied in the filling, but it necessitates an attachment to the rope that is quickly made and unmade, while it is yet entirely secure. A spring hook is used for this purpose, such as shown in Fig. 921.

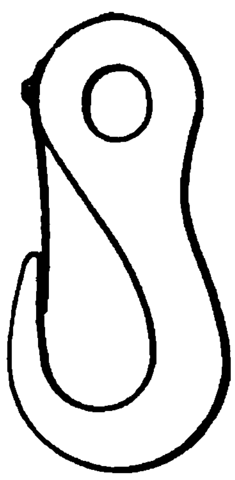


FIG. 921.

Sometimes the bucket is suspended from a ring by three chains, instead of the handle shown, and in that case the loaded bucket at the top is replaced by an empty one, the loaded bucket being taken away to be emptied.

2535. In cases where the shaft is sunk to a considerable depth, and there is a likelihood of the bucket swinging about in the shaft, a **cross-head** is used to guide it. This is simply a frame, usually of wrought iron, fitted to run up and down on the guides, and arranged to take hold of the end of the rope just above the bucket. Such a cross-head is shown in Fig. 922, in which *a* is a side elevation, *b* a plan of the lower cross-piece, and *c* a plan of the upper cross-piece. The guides are shown at *A* and *A*. The cross-head consists of an upper cross-piece *B* and a lower cross-piece *C*, tied to each other by the angle-irons *D* and *E*. At each end of each cross-piece there is a shoe which fits over the guide easily, so that it can slide up and down. These four shoes, therefore, control the motion of the cross-head. The upper cross-piece is a vertical plate fastened to the shoes by angle-irons, and bent in the middle to clear the

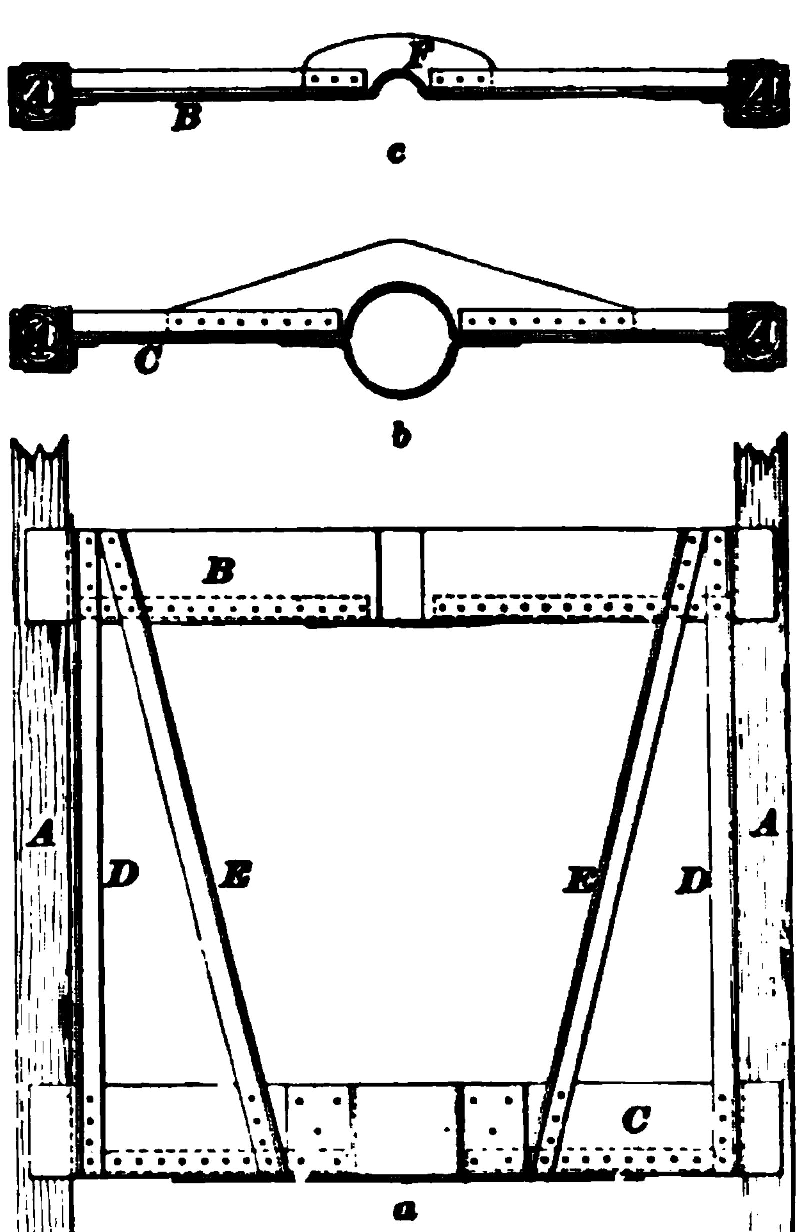


FIG. 922.

rope. It is stiffened transversely by angle-irons along its lower edge, but these do not pass the center because of the offset for the rope. To stiffen it here, a horizontal plate is riveted to the under side of the angle-irons, as shown at *F* in the plan *c*. The lower cross-piece, likewise, is a vertical plate, fastened to the shoes and stiffened in the same manner as the upper cross-piece. At the middle, however, it is bent out into a semicircle of considerable size, and to it is bolted another plate or strap, also bent into a semicircle. This forms a ring, into which is placed the box, shown in Fig. 923 to a larger scale. The box has flanges outside and inside, top and bottom. Its diameter *A* is the same as the inside diameter of the ring of the lower cross-piece, and the distance *B* between its outside flanges is the same as the depth of the cross-piece. In other words, it is made to fit the ring, and becomes a part of the cross-head when in place, with the strap bolted over it. It is lined with wood, and both it and the wooden lining are in halves, so as to get them over the rope. The rope has a cone on its end—sometimes the rope-socket is made to answer this purpose—and this cone fits in the conical hole in the wooden lining.

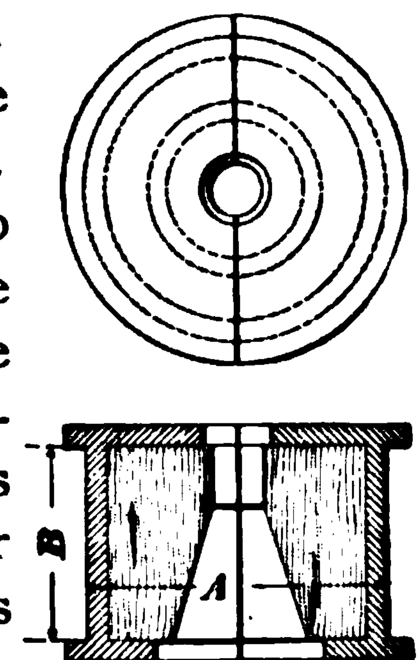


FIG. 923.

The cross-head then rests on this cone and is carried up and down by the rope, which it in turn guides and steadies. In sinking, there is always more or less distance from where the guides end down to where the loose rock is. Stops are, therefore, put on the ends of the guides to take the cross-head when it comes down, and to allow the bucket to go on to the bottom. When the bucket comes up again, the cross-head is picked up and carried to the top, during which time it keeps the bucket from swinging about.

2536. Occasionally, when the hoistway from the mine is a slope, and especially when it approaches the horizontal, the ordinary mine-cars are used to convey the material to the surface. They need not differ, however, from the

ordinary car because of this service, and we will, therefore. not take them up here. They are not essentially a part of the hoisting plant, and so do not enter our present studies.

CAGES.

2537. The surface conditions of a hoisting plant are often such as to necessitate the carrying of the material some distance from the mouth of the hoistway. It is then advisable to bring the mine-cars to the surface with their contents, in order to avoid the unloading and reloading of the material; and this, too, when the hoistway is a vertical shaft or a slope that is too steep to run the mine-cars on. In such a case an auxiliary car is used to carry the mine-cars; and this auxiliary car is known as a **cage,** whether it is for a slope or a shaft.

When the inclination of the slope exceeds 35°, the cars are usually raised on cages, because the material will not stay in them on steeper pitches. These cages are sometimes built to run on a slope-track as arranged for mine-cars, but, to insure stability, they are generally of a broader gauge. The head-room necessary is governed not so much by the form of cage as by the length of the car and the inclination of the seam. This height is less when the cars are placed on the cage with their length across the slope than when they are run on lengthwise; but as this arrangement increases the width of the slope, it is not always an improvement on the other method. When the inclination of the vein is very steep, the wheels are sometimes placed on the sides of the cage and above the center of gravity, and are run on tracks or guides supported by timbers on each side of the slope. Such slopes resemble shafts in many particulars, and in metalliferous mining districts would be so described

2538. The cage shown in Fig. 924 is a good stiff cage built of wood, simple of construction and easy to repair. Its details will be readily understood from the illustration, except, perhaps, the device for locking the car to prevent its running off during the hoist. The platform *A* having a

piece of the car track on it, may move vertically up or down. As shown in the side elevation, it is resting on the horizontal timbers *B* of the cage in a position ready for hoisting. At the end of the hoist, when the cage settles upon the keeps *C* and *C*, shown in the end elevation, this platform reaches them first and is supported by them while the rest of the cage descends still farther, until the timbers *D* and *D* rest upon the keeps also. The track on the plat-

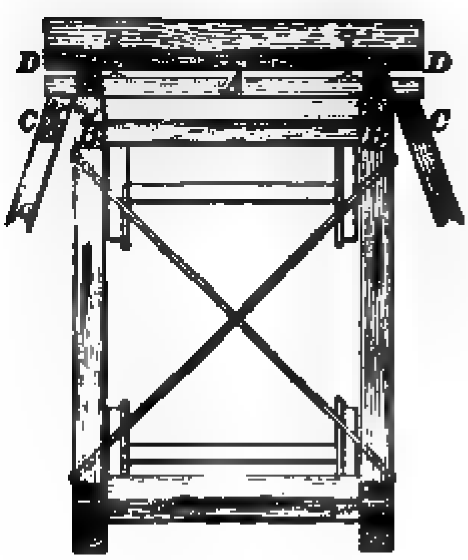


FIG. 924.

form *A* is then at the same level as that on *D*, and the car can be run off and replaced by another. When the new car is on and we are ready to make another trip, the cage is lifted from the keeps, but the platform remains until the timbers *B* and *B* of the cage come up and pick it up. Then the keeps are swung back out of the way, and the descent is made. Keeps for such a purpose must be part of every cage, whether for slope or shaft.

2539. Fig. 925 shows a **shaft cage** of modern construction built of wood. Besides the cage proper, it shows

several appliances that should be common to all cages in some form or other. In the first place, there is a covering A and A at the top, called a **canopy,** or **bonnet,** to protect any person on the cage from objects falling down the shaft. This is shown as made of heavy wrought-iron plate,

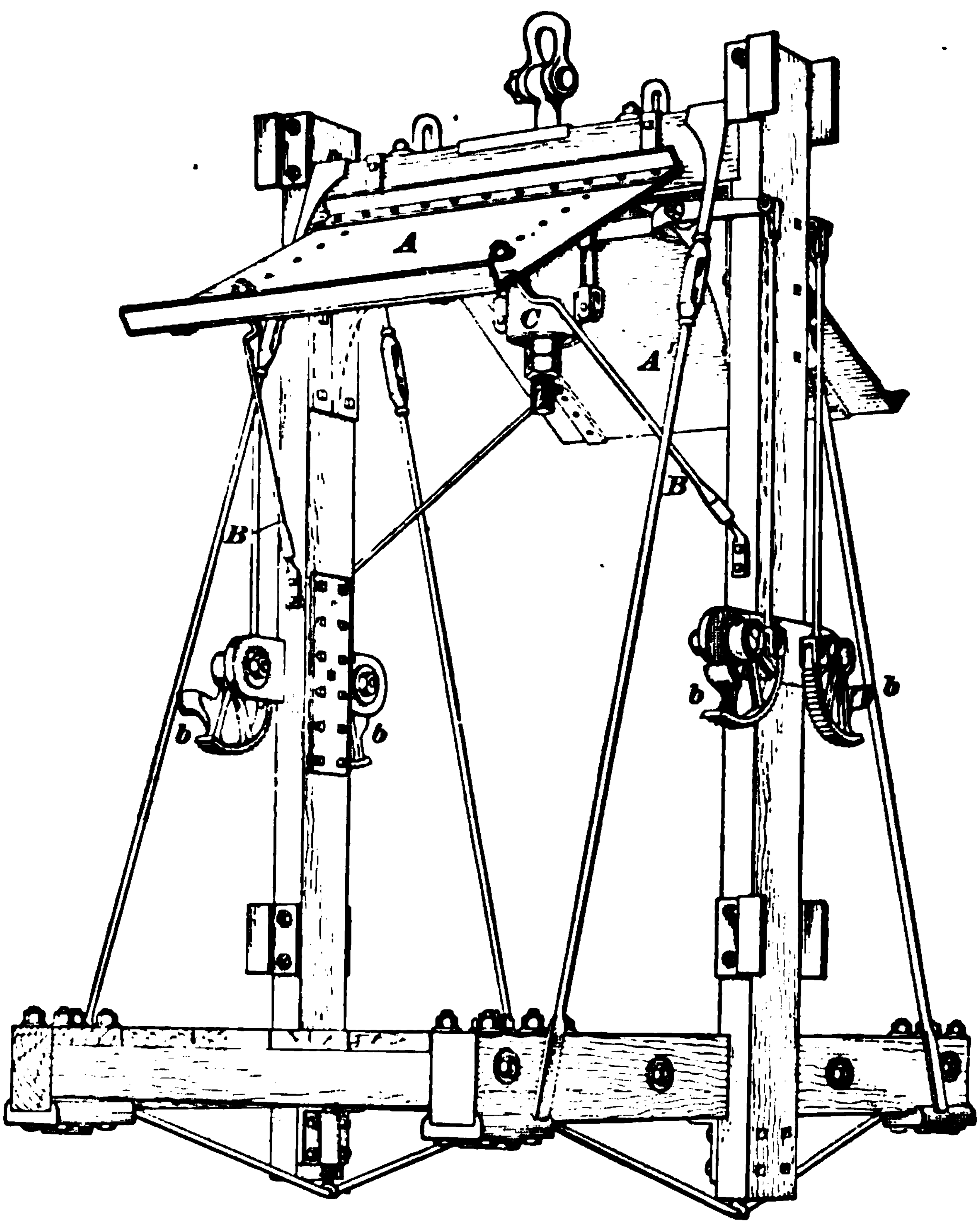


FIG. 925.

with flanges or angle-irons to stiffen it. Sometimes planking is used for this purpose. These canopies are required by law in the State of Pennsylvania. They are usually inclined, so that objects falling upon them will slide down and drop upon the cage. To prevent objects of moderate size

from wedging between the edge of the canopy and the shaft lining, they are usually made shorter than the cage, so that a space of about a foot is left between the lower edge of the canopy and the shaft lining or buntons. One part of the canopy, as *A*, is fastened to the cage by two hinges and held up by two rods *B* and *B*, which have sockets at their lower ends, and which fit over pieces bolted to the uprights of the cage. This is done so that long timbers for props and other purposes may be lowered on the cage very readily.

2540. Besides the canopy, as shown in the figure, there are also **safety-catches,** consisting of a pair of toothed cams *b* and *b*, located on either side of the cage, near the guides, and arranged to be thrown in against the guides by the spring *C*, in case the rope breaks.

The law of the State of Pennsylvania requires an improved safety-catch on every carriage used for lowering or hoisting persons.

In Fig. 926 the safety-catches are shown to a larger scale. The upper head-piece, or cross-piece, of the cage is shown at *A*. The draw-bar *B*, to which the rope is attached, runs through this and extends far enough down to carry a plate *C* with a spring between it and the head-piece. The weight of the cage, therefore, is carried by the head-piece through the spring on the plate *C*. The spring shown in the figure consists of three pieces of rubber with cast-iron plates in between them, to guide them and help them keep their form under pressure. Rubber springs, however, are not essential. In this same design of safety-catch, a helical steel spring could be used instead of the rubber; or, with a little different design, a flat steel spring could be used. Around the spring is shown a cylinder *D*, which serves two purposes: it protects the spring from the moisture in the shaft and also serves as a guide to the plate *C* by being cut away to receive it at *E* and *E*. To the plate *C* are attached links *F*, *F*, which in turn take hold of the levers *G*, *G*. A plan of one of these levers is shown at *H*. From the outer ends of the levers, connections *K*, *K* extend down to and

are attached to the eccentric-shaped toothed brass cams, or dogs, *J*, *J*.

Let us now follow out the action that takes place from the draw-bar to the cams at the left. The draw-bar is shown in its lower position, being pushed down by the rubber spring and not held up by the pull of the rope. It has carried down with it the plate *C*, the links *F* and *F*, and the inner ends of the levers *G* and *G*. The outer ends of the levers are, therefore, in their upper position, as are also the connections *K* and *K*; and this puts the cams, or dogs, into

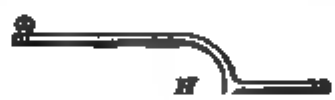


FIG. [illegible].

such a position that their first teeth will engage with the guides. The instant this occurs, if the cage is falling, the cams will be carried around, digging their teeth deeper and deeper into the guides, and thereby stopping its fall. The cams are provided with a projection *a*, which strikes the guides and prevents them from turning all around.

When the cage is picked up by the rope, the spring on the draw-bar is compressed, allowing the draw-bar to take its upper position as shown by the dotted circles. This carries up the plate *C*, the links *F* and *F*, the inner ends of the

levers G and G, and causes the outer ends of the levers and their connections K and K to move down, thus rotating the cams away from the guides. This position of the cams is shown at L.

Some engineers prefer placing the cams, or dogs, nearly opposite the cage platform, while others place them near the top of the cage. The latter arrangement seems preferable, because it simplifies the construction.

2541. One of the chief difficulties that has been experienced in the use of safety-catches is their tendency to be thrown into action by any sharp jars or shocks to which the cages may be subjected, by the sudden stopping of the engine or by the landing of the cage at the top or bottom. But accidents from this cause are now extremely rare; and if proper care is taken in adjusting the springs and catches, there seems to be no reason why any such difficulty should occur. Practical tests of the catches in use, made by hanging the cage and allowing it to drop, show that they are, as a rule, very efficient devices. The cams usually take hold at once, the cage dropping only a few inches, or, at most, a few feet. When the guides are very greasy or wet, the trial tests are sometimes not so satisfactory, and the cage may drop several feet before the cams take a firm hold and stop it. The results are still less satisfactory when the guides are covered with ice; but even in this latter case, the cage sometimes drops less than one foot.

Fortunately for the utility of safety-catches, ropes are usually broken while a loaded cage is being raised, and the cage has an upward momentum; if a rope breaks on the empty side, and when the cage is rapidly descending, at a speed, say, of 30 ft. or 40 ft. a second, its momentum is so great that either the catches must break or the cage or guides and shaft lining will be torn to pieces. The catches generally hold, and either the guides or cage suffer more or less injury under such circumstances. In experimental tests, with ice-covered guides, the cage has been known to fall eight to fifteen feet before the cams plowed their way

through the ice and took firm hold of the guides; but the momentum the cage acquired was so great that the guides were destroyed.

2542. Both of the cages already illustrated are built principally of wood. This is not the universal practice, for iron and steel cages are also largely used. In Fig. 927

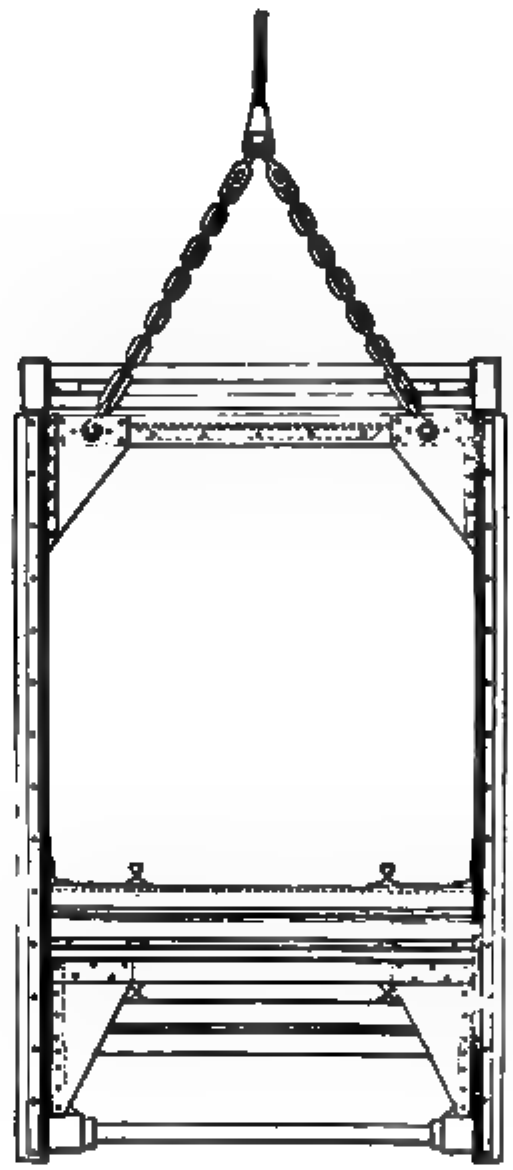


FIG. 927.

is shown an iron cage which also illustrates another feature; that is, its ability to dump The figure shows the cage partly dumped. The mine-cars are run on to this cage at the bottom, just as they are on any cage, except that they are more thoroughly secured, and are hoisted into the head-house and dumped while still on the cage. As will be seen,

the cage consists of two parts. The first, a frame *A* made up of the upper cross-piece, to which the rope is fastened, two guide pieces, and a distance-piece at the lower end; the second, the cage *B* which rests upon trunnions secured to the frame *A*. The uprights of the cage carry dumping wheels *a* on each side, which run in special tracks *b* made for them in the head-frame, and so turn the platform of the cage over to the dumping angle. The use of dumping cages is principally limited to cages used for simply raising mine-cars from the surface to the top of the breaker.

Few engineers consider the use of dumping cages for winding shafts advisable or even allowable. They are, at best, cumbersome, more or less complicated, and, consequently, liable to accident.

GUNBOATS, OR SKIPS.

2543. Gunboats, or skips, are self-dumping cars used for hoisting in a slope. They are usually built of boiler-iron, although sometimes of wood, and are generally large enough to hold several mine-car loads; that is, they often carry four to five tons of coal. A scheme peculiar to gunboats is to have the wheels fixed upon the axles and the axles seated in pedestals. This insures smoother running than is obtained with loose wheels, but the pedestals are so located that they are not readily accessible for lubrication.

2544. Fig. 928 represents a modern gunboat, closed along the top *a* and open at the end *b*, which is cut about the angle of the slope in which it is to be used. It will be noticed that the bottom, sides, and top are stiffened by angle or tee irons, and the back is stiffened and protected by 3-inch planks, backed by 3 in. × 6 in. timbers. The details of the pedestal bearings are shown in Fig. 929. They consist of three castings: the pedestal, or bracket, *a*, which is bolted or riveted to the gunboat; a pivot casting *b*, and the bearing proper *c*. The bearing

rests upon the axle and carries, by means of trunnions *d*, the pivot casting *b*, on the top of which is placed a

FIG. 998.

rubber cushion *e* to lessen the shocks between it and the bracket.

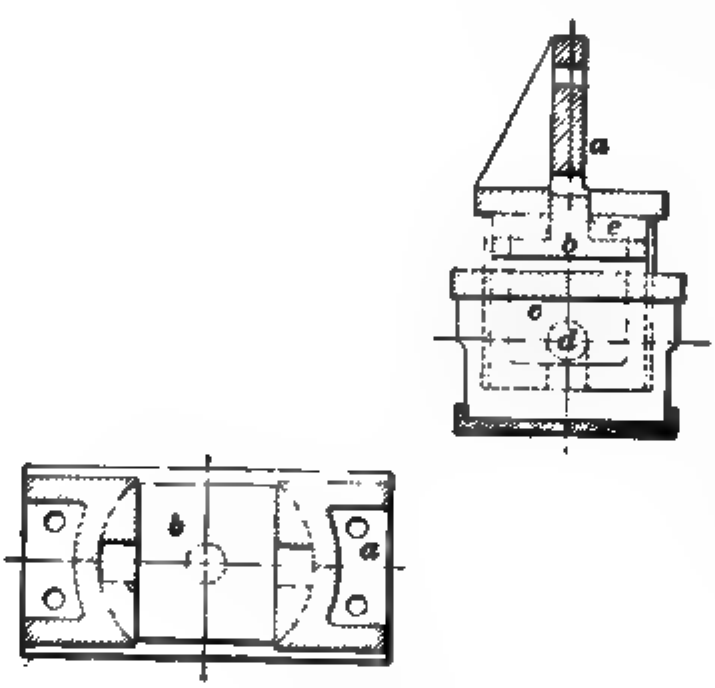


FIG. 999.

The manner of dumping a gunboat is explained under the head of dumps.

One objection that has been claimed against the use of gunboats is that, with coal, the dumping of the coal from the mine-cars into the gunboats causes more breakage and,

therefore, more culm. This, of course, would not affect their use at any of the metal mines.

They have the advantage of being attached to the rope all the time, thus saving the time necessary to hook and unhook the mine-cars.

2545. Fig. 930 shows a gunboat *a* in a slope, and standing immediately below a level where there is a car *b* ready to have its load dumped into the gunboat.

FIG. 930.

ROPE CARRIERS.

INTRODUCTION.

2546. In a hoisting arrangement of the utmost simplicity, such as we would have if we used a windlass, like that shown in Fig. 888, placed directly over the hoistway, there would be no necessity for a rope carrier, for the rope could hang from the drum naturally. If, however, the drum had to be of a considerable length, or the winding edge of it had to be located anywhere but over the center of the hoistway, we would find it necessary to use **rope carriers** to guide and control the rope in its passage from the drum to this center line. This is the case in all hoists except the smallest and most primitive. In many cases the rope carriers of a hoisting plant require much study to properly design them.

A rope carrier, generally speaking, is a wheel supported in a frame so that it can revolve and allow a rope to run over it, and its use is to enable us to deflect the rope from a straight line or to keep it in a straight line against the action of gravity. Many head-frames are simply rope carriers; but

as most of them serve other purposes, we will take them up in detail under the head of tracks.

We will consider here only *sheaves* and *rollers*, which are used as rope carriers directly.

SHEAVES.

2547. Sheaves are grooved wheels used to carry or guide the hoisting rope. It was formerly a common practice to build them up with wooden arms and rim on a cast-iron hub, and many such sheaves are in use to-day. A good feature of such sheaves is their light weight, which allows them to be put in motion and stopped readily by the rope running over them; but they are not durable. Unless they are built up of many pieces, they will not keep their shape, and if they are built up of many pieces they are costly. They become soaked with oil from the rope and from the bearings in which they run, and are, therefore, a source of danger from fire. They also look clumsy, and are unmechanical.

2548. The sheaves of to-day are built entirely of iron and are of two styles—those composed entirely of cast iron,

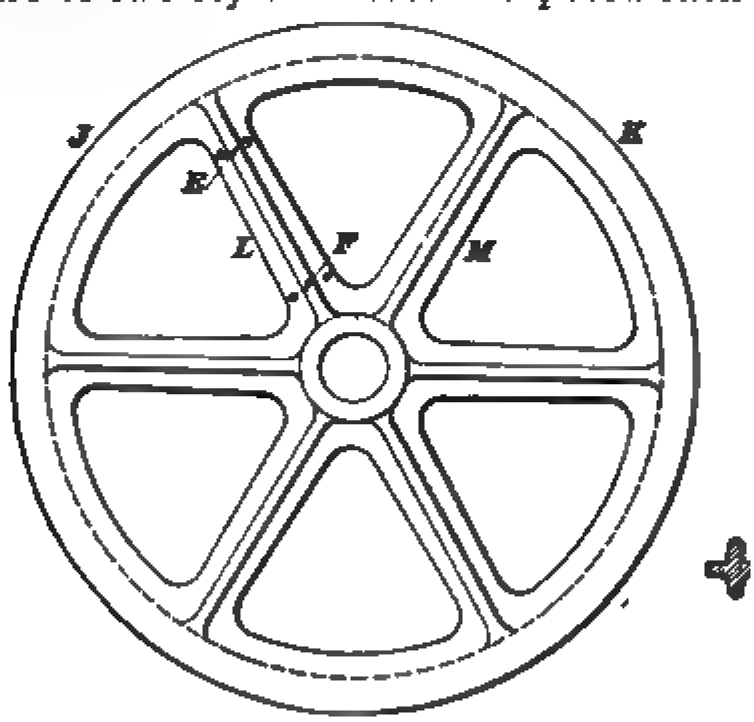


FIG 981

and those with cast-iron hubs and rims and wrought-iron arms, or spokes. An illustration of the first style is given

in Fig. 931. This sheave is the cheaper of the two styles, and for many purposes it is entirely satisfactory. Its great weight is an objection, because it adds to the weight on the journals, and also offers considerable resistance to being set in motion. If such a sheave is used to carry the rope in a straight line or to deflect it only a little, the pressure of contact between the rope and the sheave will be slight and, therefore, also the ability of the rope to turn the sheave. In such a case, when the rope starts or stops quickly, as it is very likely to do in modern hoisting plants, the sheave lags behind, and the consequent slipping is a great source of wear.

The arms have a cross-section as shown at *A B*, a form that is very stiff in every direction and easy to mold. They should taper in both directions; that is, dimension *C* should be less than *D*, and *E* less than *F*. This gives a lighter and more shapely wheel for the same strength. The bottom of the groove *G* in the rim should be a circular arc whose radius is a little larger than that of the rope, to allow for the angle of the rope due to its fleeting on the drum. The flanges at *H* are added to prevent the rope from jumping off if anything gets under it on the sheave.

2549. A sheave with cast-iron hub and rim and wrought-iron spokes is shown in Fig. 932. This is a very excellent

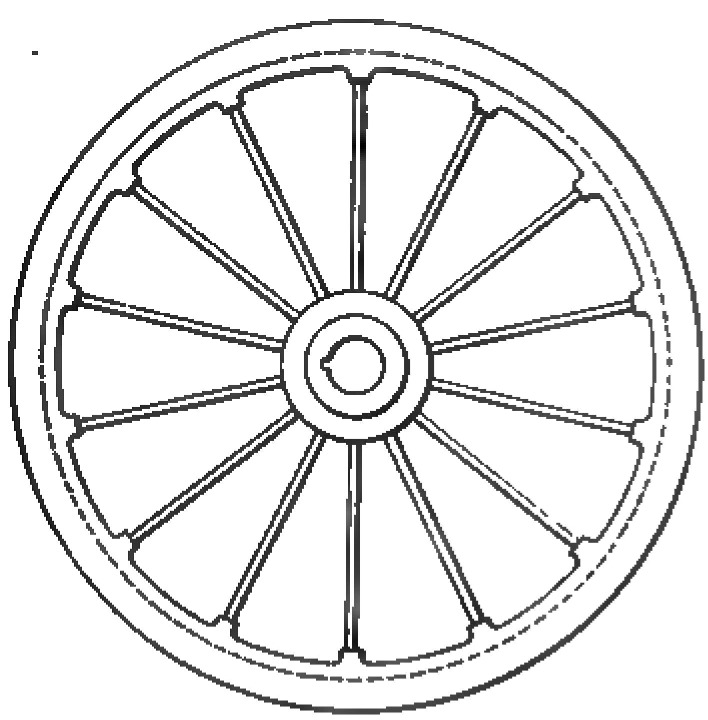


FIG. 932.

sheave, especially for the larger diameters. It is extensively used both in the United States and abroad.

The principle of construction of such a sheave is very different from that of sheaves entirely of cast iron. In the case of a sheave with cast-iron arms, the load put upon it by the rope is transmitted to the shaft by a compressive stress through those arms directly under the load. In other words, if a rope is run over the top of the sheave in Fig. 931, putting a load upon it, say from *J* to *K*, this load will be transmitted as a compressive stress through the arms *L* and *M* to the hub and the shaft. Of course, we can readily see that a part of it might be carried around the rim to the lower arms, and be supported by them in tension; but we should not consider that in designing the wheel, because cast iron is of comparatively little value in tension, whereas it is of great value in compression.

Now, in the case of a sheave with wrought-iron arms, or spokes, as shown in Fig. 932, the load would be transmitted around the rim to the side opposite its point of application, and carried from there to the hub by the tension of the adjacent spokes. In fact, the spokes all around the wheel are in tension continually from the method of construction, and the sheave is very strong and rigid. This gives a very light wheel for a required strength, and there is, consequently, little wear between it and the rope due to any slipping action when it is started or stopped. As will be seen in the sectional view of Fig. 932, the spokes are carried to the right and to the left alternately, so as to take hold of the opposite ends of the hub, thereby giving stiffness to the sheave against any side stress.

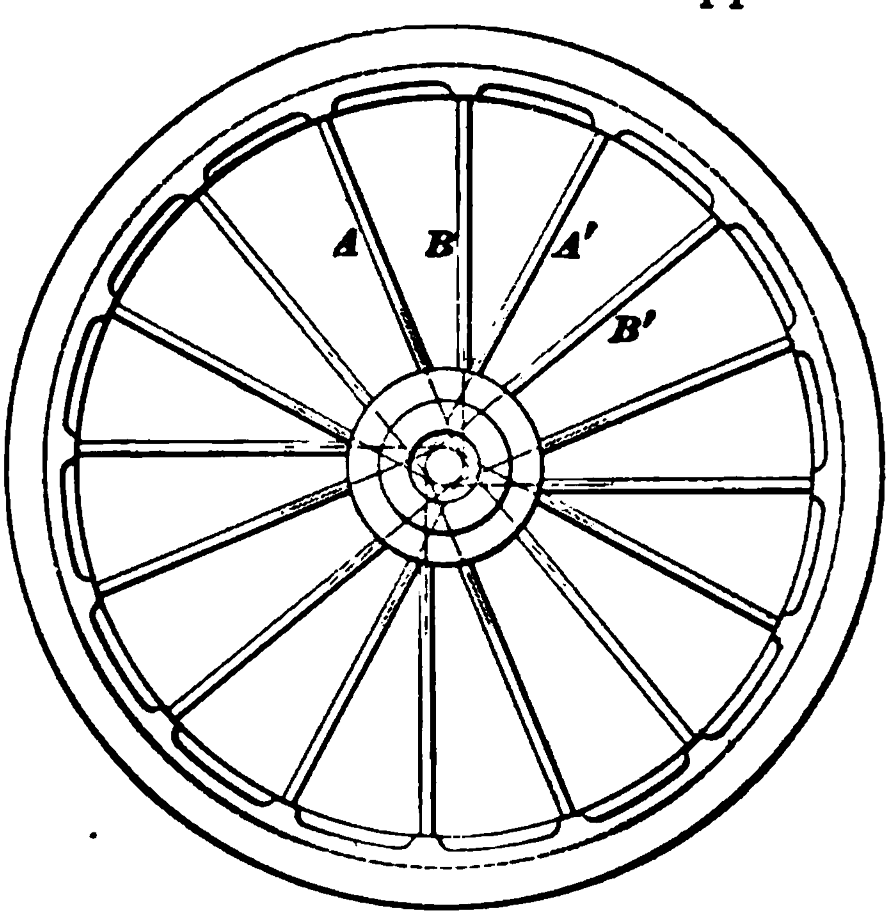


FIG. 933.

Sometimes, also, the spokes are put in slightly tangential instead of radial, which method is shown in Fig. 933. In the center of the wheel there is an imaginary circle, say two inches in diameter for a 10-foot sheave, and to this circle all the spokes are tangent. Taken in pairs, made up of alternate spokes, they are made tangent to the opposite sides of the circle, so that they pull against each other, so to speak, and thereby make the sheave rigid in both directions. The alternate pairs are carried to the two ends of the hub to give lateral stiffness. In other words, spoke A is tangent to the right side of the circle, and spoke A' is tangent to the left side. These pull in opposite directions, so far as their tangential pull is concerned. Then, spoke B is tangent to the right side of the circle, and spoke B' is tangent to the left side; but this pair B and B' take hold of the lower end of the hub, while the pair A and A' take hold of the upper end. This arranges the spokes in groups of four, so the total number must be some multiple of four. The tangential direction of the spokes is often necessary in very large sheaves carrying heavy loads, because it requires a very considerable force to turn the shaft in its bearings. With radial spokes, we have only their strength, or rather stiffness, considered as beams to do this turning; but with the tangential spokes we have a direct pull to do it. This is the best sheave that we have to-day.

2550. The rims of both of the above styles of sheaves are made either solid or with wooden lagging, as shown in Fig. 934, which is a section through such a rim. The left flange is a separate piece, which is centered with the sheave by the lip at A, and which is held on by bolts, as shown at B. The blocks of wood have holes bored in them for these bolts to pass through, and are held securely by the clamping together of the two flanges with the bolts. With such a sheave, there is much less wear of the rope than there is with one that has a plain cast-iron rim.

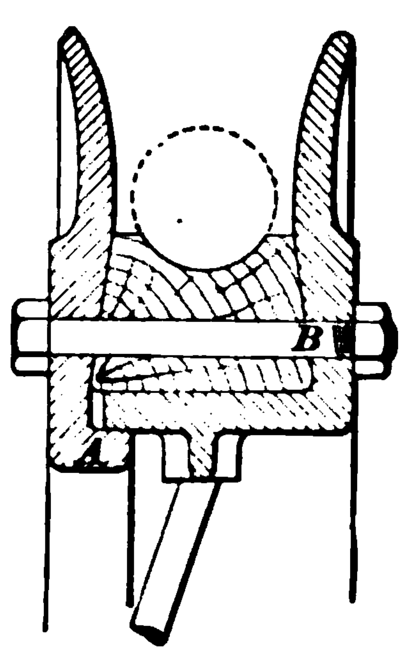


FIG. 934.

The wear of the sheave proper is also avoided, because, as the blocks wear down, they are taken out and replaced by new ones.

The size of a sheave for a given purpose depends, generally, upon the size of the rope to be used, but if the rope is simply to be carried in a straight line against the action of gravity, it may depend upon other circumstances. In such a case the usual method would be to use a roller. If the rope is required to bend over the sheave, as it is when its direction is changed, the sheave should bear some relation to the diameter of the rope. In Mine Haulage, it is stated that the minimum diameter of drum or sheave over which a wire rope having 19 or 7 wires to a strand is to be led is 60 and 100 times the diameter of the rope, respectively. But we should not use this minimum diameter unless we are compelled to, for the larger the sheave the less will be the wear of the rope due to the bending, and, consequently, the longer the life of the rope. Of course, the cost of the sheave, which necessarily increases with the size, would put a limit in the other direction.

ROLLERS, OR CARRYING SHEAVES.

2551. **Rollers** are used for rope carriers when the only object in view is to sustain the rope against the attraction of gravity. There is no bending of the rope in this case, except a very slight amount due to the sagging between the rollers, so the diameter of the roller is not of any importance, so far as the rope is concerned. If the rollers are for use on a slope to keep the rope from dragging on the ground, it is evident that they must be small, because the car must run over them; and mine-cars are usually made low because of restricted head-room in the mine.

Let it be understood, however, that if the rope is required to change its course from a straight line, a roller will not answer, but a sheave must be used, even if the deflection is only a small amount. We have just said that where rollers are used to carry the rope against the attraction of gravity, there is no bending of the rope, except a small amount due

to the sag. Now, this small amount of bending is not harmful, because it does not produce a stress. When the load is put upon the rope, its tendency is to take a straight line and decrease the bending. On the other hand, if the rope turns a corner or is deflected from a straight line when the load is put upon it, it tends to hug the turning sheave and to bend as sharp as the sheave will let it. Therefore, in such a case, a sheave should be used in proportion to the size of the rope.

TRACKS.

CLASSIFICATION.

2552. In all hoisting operations, except in very shallow vertical hoists, it is necessary to have some sort of a track to guide and control the car or cage during its travel. If the direction of the hoist is other than vertical, that is, if the hoistway is a slope, then the tracks are similar in most respects to those of any other haulage way where tracks are used. If the hoist is a vertical one, the tracks must be different, and they are then called **guides,** or **conductors.** In either case, it is often found advisable to change the tracks or guides into dumps at the upper ends of the hoists, which empty the car automatically. There are also located at the different landings, **keeps,** or **landing fans,** on which to rest the car while it is being loaded. We would then have under this head the following items to consider; namely, *Car tracks; Guides, or Conductors; Dumps; Landing fans, or Keeps; Head-frames.*

CAR TRACKS.

2553. Tracks such as we see on any ordinary railroad, are used to run the cars on while they are being hoisted up a slope or inclined shaft. These tracks are generally of narrower gauge and of lighter construction than most surface railways, but they are similar. They are usually of **T** rails, varying in size from the lightest pattern up to 60 lb. per yard, laid on cross-ties in the ordinary way.

When the regular mine-car is hoisted, the track must be provided with turnouts and switches, so that the cars can readily be run from the slope into the gangway or level, and *vice versa.* These gangways are generally run at right angles to the slope, and may be anywhere in the length of the slope as well as at the bottom. At the foot of a slope, or at the landing on any lift, the gangway is widened out to accommodate two tracks, one for the empty and one for the loaded cars. The empty track is generally on the upper

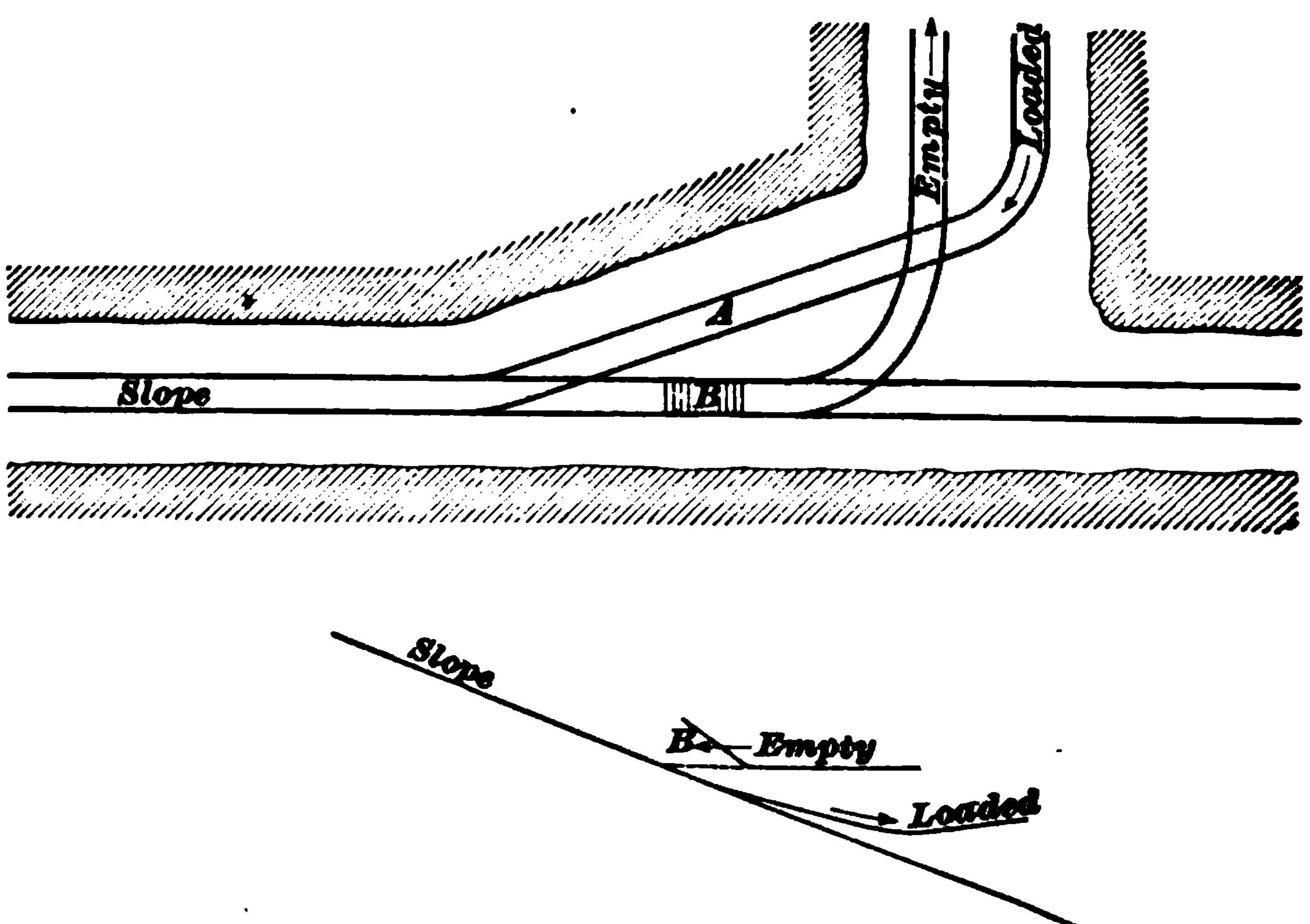


FIG. 935.

side of the gangway and the loaded track on the lower side. One method of arranging the tracks is shown in Fig. 935.

At a distance of forty or fifty feet above the gangway, the slope is widened out to accommodate the branch *A* which connects with the loaded track. It will be noticed that the tracks in the gangway are not on the same level near the slope. The empty track is kept high, so that the empty cars will run by gravity into the gangway; and the loaded track is dropped down, so that the loaded cars will run by gravity from the gangway on the branch *A* ready for hoisting. A short distance above the gangway, at *B*, a bridge or door is placed which, when down, forms part of the empty track

and switches the empty cars coming down the slope off into the gangway. This bridge or door is shown to be open in the lower part of the figure at *B*. The empty track is about six feet higer than the loaded track and is carried over it on a trestle. When this door is up, the track is continuous down the slope to a lower gangway or to the bottom. The illustration shows the plan arranged for a single slope fed from one side, or half of a double slope fed from both sides.

When hoisting is being done from this level, the bridge is down. The empty car comes down and is run off over the bridge, the car is unhooked from the rope, and the end of the rope is thrown down to the branch below, where a loaded car is standing. This car is attached to the rope, and is hoisted up the branch and through the switch to the main track.

2554. This plan can be employed economically only in thick seams, as the height necessary to allow one track to cross

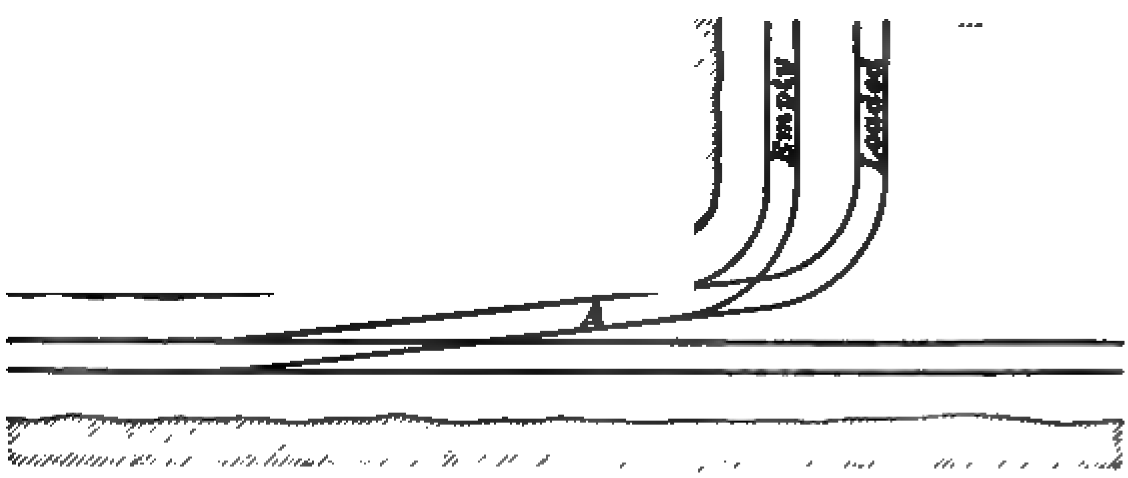


FIG. 936.

the other on a trestle can not be obtained in seams of modern thickness without taking down a large amount of top. A simpler plan, which does away with the bridge, is often used. This is shown in Fig. 936, in which the branch *A* is laid out as before, but near the point where it enters the gangway a switch is placed opening into the empty track. The disadvantage of this arrangement is that the cars can not be handled by gravity as before.

2555. A common arrangement of the tracks at the bottom of a slope is shown by Fig. 937. A branch is made by widening out the slope near the bottom, and this being a few feet higher than the main track, is used to run off the empty cars by gravity. The loaded cars run in by gravity around the curve to the foot of the slope, in position to be attached to the rope. In ascending, the loaded car forces

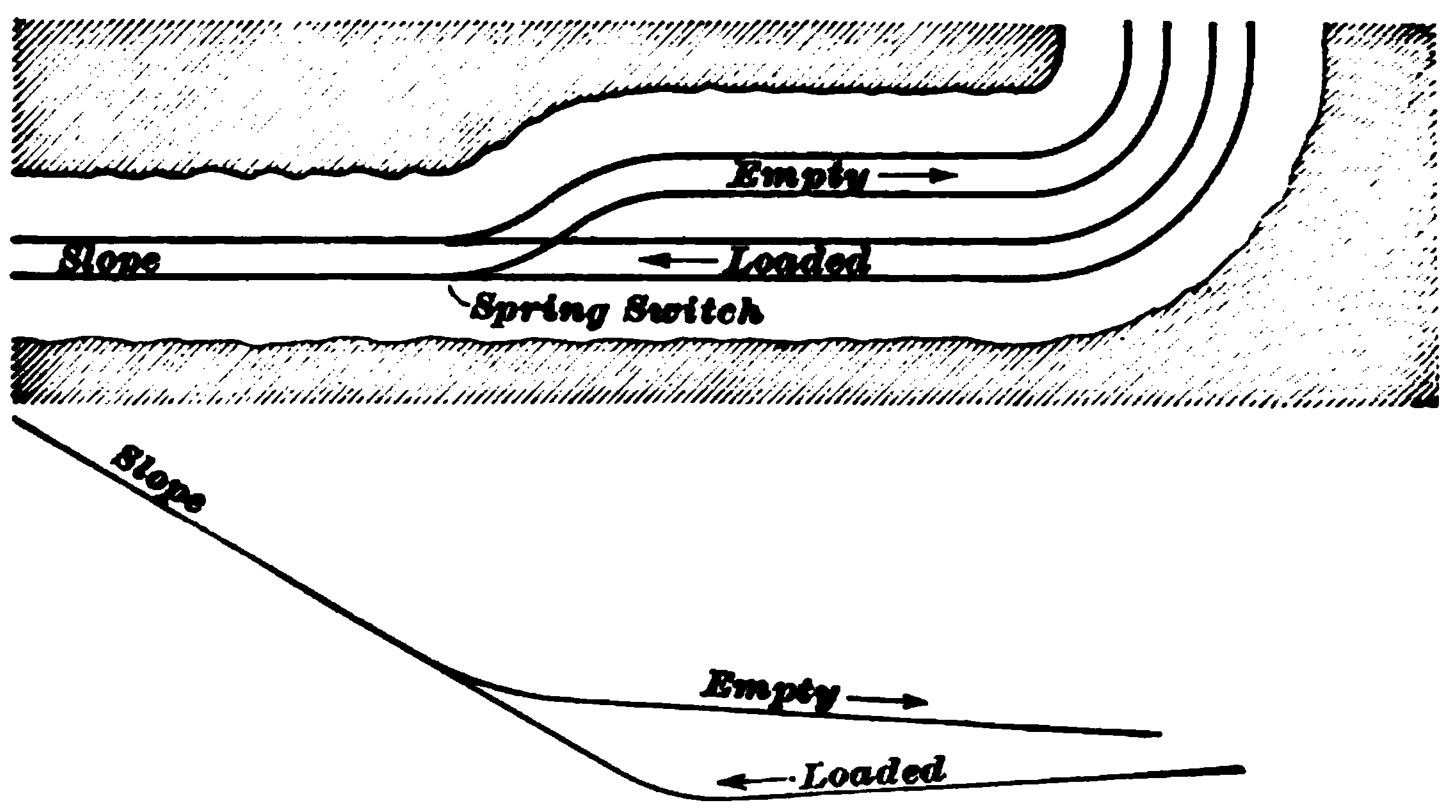


FIG. 937.

its way through the spring switch, or the switch may be set by a lever located at the foot of the slope. When the empty car descends, it runs in on the branch, where the rope is unhooked and thrown over in front of the loaded car; the car then runs around the curve into the gangway by gravity.

GUIDES, OR CONDUCTORS.

2556. Guides, or **conductors,** are used in all vertical shafts of any considerable depth, to serve as a track to keep the car or cage from swinging about and striking the sides of the shaft. They are made of wooden rails, iron rails, or wire ropes. In English mines, wire ropes are used to the exclusion of almost everything else, but in American mines wooden guides predominate, although some iron ones are used. Probably this difference in practice is due to the different shapes of shafts that are used in the two countries.

In English mines, the usual shape of cross-section of the hoistways is circular, while the cages are rectangular, and the same opening is used for ventilating. In such a case, the wire-rope conductors leave the shaft practically unobstructed, because they simply hang there without any bracing or tying transversely. In American mines, the usual shape of cross-section is rectangular, as is also that of the cage, and there is only a working clearance between them Wooden guides seem more suitable in such a case.

2557. There should be three or four conductors for each cage where wire ropes are used for the purpose, because they sometimes break. If only two are used and one breaks, the remaining one can not control the cage, but will let it swing about itself. If three or four are used and one breaks, there will still be two or three left, and they will, of course, keep the cage where it belongs. In order that wire-rope conductors may give a positive guidance to the cages, they must be under tension. This is sometimes done by drawing them taut and fastening the lower end, but this is not a good method. When the rope expands with the heat, it gets slack and is useless as a conductor; and when it contracts with the cold, it either breaks itself or tears away from one of its fastenings. The best way to do is to hang a weight to the lower end large enough to give the necessary tension, and to arrange guides so that the weight can travel up and down. If the conditions at the foot of the shaft make such a plan objectionable, the rope can be secured at its lower end and have its upper end hung on one end of a lever, on the other end of which the necessary weight is applied. Of course, this weight must be large enough to counterbalance the weight of the rope and give the tension to its lower end.

2558. There is little variation in the details of wooden guides. They are always of a rectangular cross-section. In some localities, especially in Europe, where wood is not so plentiful, they vary from 5 in. × 6 in. to 6 in. × 7 in.; but in the United States, where wood is more abundant, they

are generally between 6 in. × 8 in. and 8 in. × 10 in. The kind of wood chiefly used for guides in the United States is yellow pine, but elsewhere some of the harder woods are used.

The guides for a single cage consist of two lines of rails put together in lengths ranging from 12 ft. to 16 ft., and the butts or ends of these lengths are kept in line by being seated in notches cut in the buntons for that purpose. Fig. 938 shows a plan of a cage with the guides and buntons in their normal position. The buntons are shown at *A* and *A*, and are usually timbers of 8 in. × 10 in. or 10 in. × 12 in. cross-section; *B*, *B* are the guides in section, and *C*, *C* are the cage shoes for clasping the guides and keeping the cage in line. The guides are bolted to the buntons with bolts countersunk into them so as to be clear of the shoes.

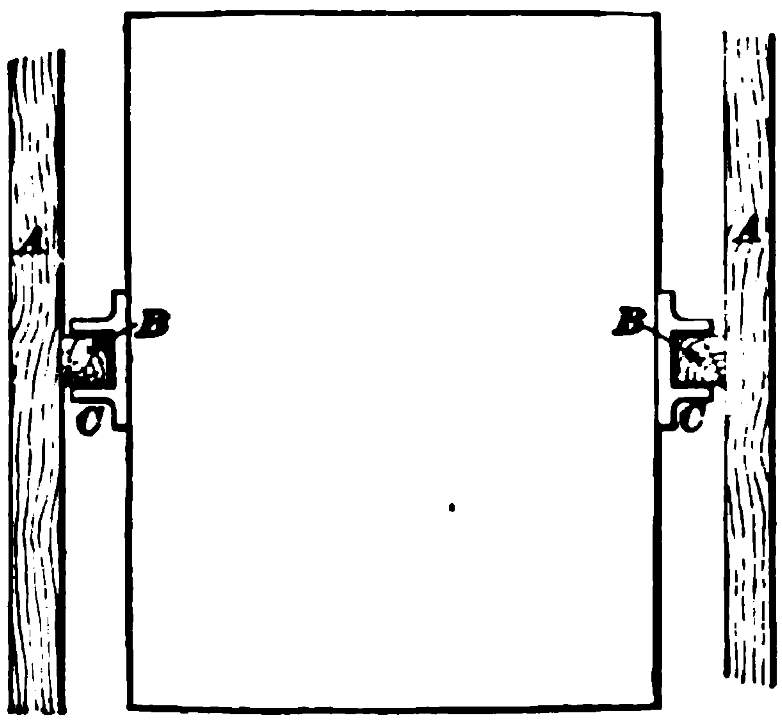


FIG. 938.

DUMPS.

2559. When gunboats, or skips, are used, the usual practice is to extend the tracks on which they run above the surface, on a head-frame, and form it into a **dump**; that is, a shape that will upset the gunboat. There is a chute to catch the material thus emptied and to conduct it into bins or elsewhere. Oftentimes, a shaft-house or head-house is built about the head-frame in which to store the material, or to prepare it for market or further handling.

These dumps are of different designs in different localities. In the Lake Superior copper region, we find many of them built as shown in Fig. 939. In this dump the rails of the main track are bent down around a smooth curve, as at *A*. At a short distance back of the commencement of this vertical curve, another track begins and runs into a straight line parallel to the angle of the hoist. This second track is of a

wider gauge than the first one, is at a higher level, and its rails are of a heavy angle-iron, as shown in section through *BB*. On the left-hand side of this section, one of the front wheels is shown, and on the right-hand side, one of the back or dumping wheels, which has a second tread of less diameter than the main tread. The second track is made of a wide gauge, in order to let the front wheels and the

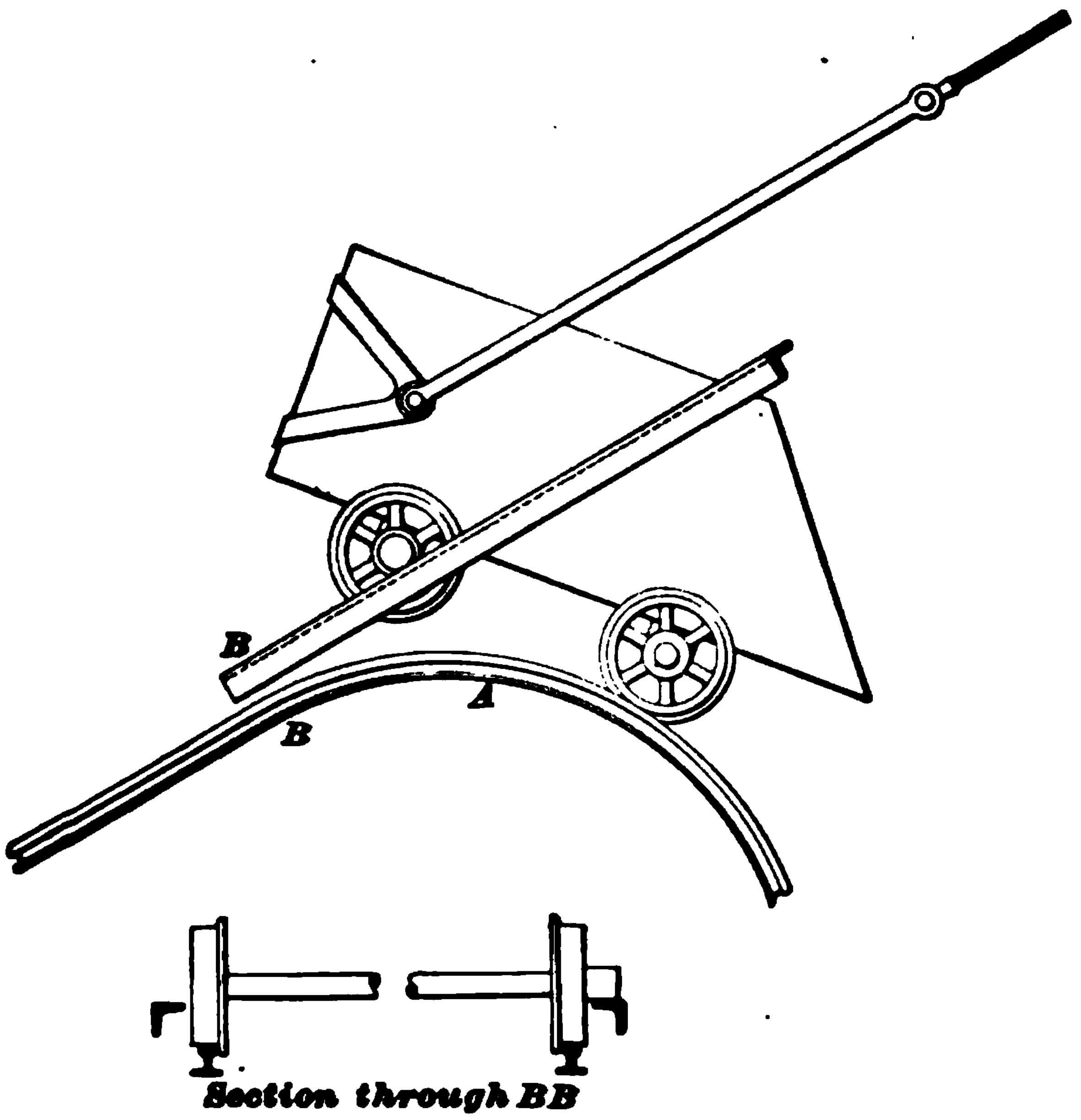


FIG. 989.

entire skip pass through it in dumping. It is raised a little above the main track, so as to reach the small tread of the dumping wheel, which runs on it. By making the dumping tread of a smaller diameter than the main tread, the latter acts as an inside flange to keep the skip from running off the track. Now, when the skip comes up, the front wheels continue on the main track, running over to the curve *A* and down, while the dumping wheels run up on the second or outside track, thus turning the skip over. If there is any overhauling beyond the point necessary to empty the skip, it will simply run along on the upper track on the two

back wheels, hanging front end down. In actual practice, this occurs at almost every hoist, and is not found to be objectionable.

The principle of action of most dumps is the same as that of the one just described, the sizes and proportions differing with the size of the skip and the angle of the slope.

LANDING FANS, OR KEEPS.

2560. At most mine openings, whether vertical or inclined, a mechanism is placed at the top, or mouth, to rest the cage upon while the cars are being changed. These are called **landing fans,** or **keeps.** A common form is shown in Fig. 940, in which *A* and *A* are a pair of cast-iron arms,

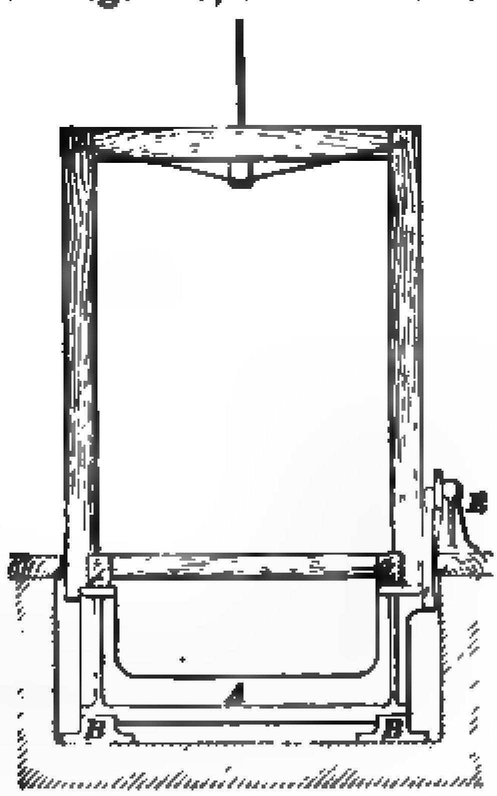


FIG. 940.

or struts, located one at each end of the cage. They rest in bearings *B*, *B* bolted to the shaft timbers, and can stand in a vertical position, entirely free of the shaft, or extend out into the shaft and under the cage, as shown. At their upper end there is an eye-bolt, corresponding to another one opposite in the timber, and between these a chain *a* is put as a safeguard to prevent the keeps from leaning out into the shaft too far or from falling into the shaft

if anything goes wrong. At the upper end of each keep there is also a pin, which extends to the side key and the line of the cage, and from these pins connections *C* and *C* extend to the center and upwards, as shown, to another pin. This last pin is carried by the end of a hand-lever *D* with a fulcrum at *E* on the floor of the landing. Now, it will be seen that if the outer end of the hand-lever is raised, the inner end will drop, and thus, through the connections *C* and *C*, force the keeps out from under the cage into the pockets in the timbering arranged for them. The cage has four cast-iron shoes with projections under it to receive the keeps, to take the wear from them, and to prevent them from swinging too far into the shaft. It should be noticed, also, that the insides of the keeps have no projections, and the operating mechanism is such that no harm would come if the keeps were left in the shaft by mistake and a hoist were made. The cage would open out the keeps and pass through them without any trouble. This is a requisite of good keeps.

2561. Most keeps that we find in use are built on the same principle as those just described, although the details of their construction may vary. An objection that can be raised against them is that, with large cages and heavy loads, the jar caused by letting the cage down on such a rigid support is very detrimental. One scheme that has been used successfully to overcome this trouble is to make the keeps of hydraulic cylinders, with plungers for the cage to rest upon. Such a keep is shown in Fig. 941. The cylinder has at its lower end an eye, about which it can turn on a pin, so as to close back into a recess in the shaft, out of the way of the cage, or extend out into the shaft under the cage. It is shown in this last position, and its operation is as follows: Suppose the cage, which rests in the jaw at *A*, has

Fig. 941.

just been lifted away. The spring at *B*, which is long and strong enough to push the plunger out the desired distance, then does this, and the plunger is ready to receive the cage again. When the cage settles down, it tends to push the plunger home again into the position in which it is shown in the figure. This action is resisted by water or, preferably, oil in the cylinder at *C*. At first this resistance is very slight, however, because V-shaped grooves have been cut in the plunger, as shown at *E*, and the oil can escape through these into the upper chamber *D*. As will be seen in the figure, the V grooves, which are of considerable size, at the end of the plunger, taper down to nothing, so the flow of oil through them becomes less and less until none can pass except by leakage around the plunger. This allows the plunger with its load to settle quietly to the bottom.

Landing fans are also used for slope cages in some cases, and may be of substantially the same design as those shown for shaft cages.

HEAD-FRAMES.

2562. A **head-frame** is a frame or structure built at the mouth or opening of a mine, primarily, to carry the head sheave over which the rope is conducted from the mine to the hoisting-engines. It almost universally, however, carries the track that the car runs on or the guides for cages, and we therefore class it under the general head of Tracks.

2563. In Fig. 942 is shown a head-frame in its simplest form without tracks or guides. The drum of the hoisting-engines is shown at *A*, with the rope coming from its upper side and running over the head sheave *B* down to the cage *C*. The head sheave is supported by the head-frame *D*. If we assume that the angles *E* and *F* that the two portions of the rope make with the horizontal are equal, the resultant stress on the head sheave will be vertical. We arrive at this conclusion through the principle of the parallelogram of forces. If two forces acting on a point be represented in direction and intensity by adjacent sides of

a parallelogram, their resultant will be represented by that diagonal of the parallelogram which passes through the

G

FIG. 942.

point. That is to say, if $G H$ and $G K$, Fig. 943, represent the directions and amounts of two forces acting upon

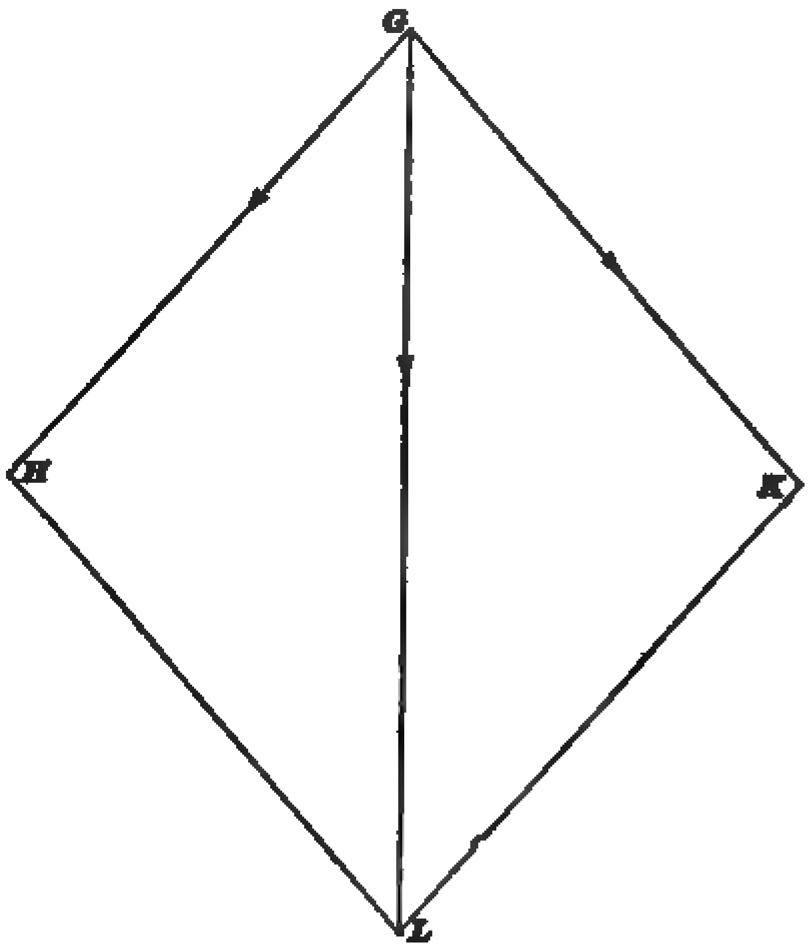


FIG. 943.

the point G, we find the resultant of these forces, or the force which would be equivalent in effect to their combined effort, by constructing the parallelogram $G\,K\,L\,H$ and drawing the diagonal $G\,L$ through G. This diagonal, then, represents in direction and intensity the resultant sought. Suppose the forces $G\,H$ and $G\,K$ are equal and amount to 20,000 lb. each, and suppose we lay off these forces with a scale representing 1,000 lb. per one-tenth of an inch. Then, the lines $G\,H$ and $G\,K$ should each be two inches long, and the line $G\,L$, which in this case measures three inches, or 30-tenths of an inch, would represent 30,000 lb. Going back now to Fig. 942, we see this operation applied to our head-frame. We have extended the rope lines to the point of intersection G, and from there have laid off the two lines $G\,H$ and $G\,K$, each one inch long, thus representing the pull of the rope, say 20,000 lb., to a scale of 2,000 lb. to one-tenth of an inch. Completing the parallelogram by drawing $H\,L$ parallel to $G\,K$ and $K\,L$ parallel to $G\,H$, and drawing the diagonal $G\,L$, we have the direction and amount of the force acting upon the head-frame in consequence of the deflection of the rope by it, while the rope is under that tension.

On measuring the diagonal, it is found to be $1\frac{1}{2}$ in. long, and as there are fifteen one-tenths in $1\frac{1}{2}$ in., and since, according to the scale we are using, each tenth equals 2,000 lb.; $2{,}000 \times 15 = 30{,}000$ lb. = the vertical stress on the head-frame. We also see from the figure that the direction of this force is vertical.

2564. Let us suppose now that we have to deal with a shaft instead of a slope, that is, with a vertical instead of an inclined hoistway. We have a representation of such a case in Fig. 944, in which, as before, A is the drum, B the head sheave, C the cage, and D the head-frame. The head-frame now has an element that it did not have before; that is, the diagonal member M. Why has this been added, and how shall we determine its location? The parallelogram of forces will answer both of these questions for us. As before, we extend the lines of the rope, which

are, of course, the lines of force with which we are dealing, until they intersect at G. From this point, lay off on these lines distances representing the stresses in the rope to any scale. For simplicity's sake, assume that the stress is 20,000 pounds, as before, and use one-tenth of an inch per 2,000 pounds for a scale. We will then have $G H$ and $G K$, each one inch long, on laying off our forces. Completing the parallelogram by drawing $H L$ parallel to $G K$ and $K L$ parallel to $G H$, and drawing the diagonal $G L$ through G, we find that the resultant is 38,000 pounds, because $G L$ is 1.9 inches long. The direction of the resultant is also determined, being in the line of the diagonal $G L$. Now, it will be seen that if we had used a head-frame for this case such as is shown in Fig. 942, it would have been overturned by this resultant force, so we added the new member M to resist that action. In all cases, then, it must be made certain that the resultant of all the forces acting on the head-frame shall fall within the structure.

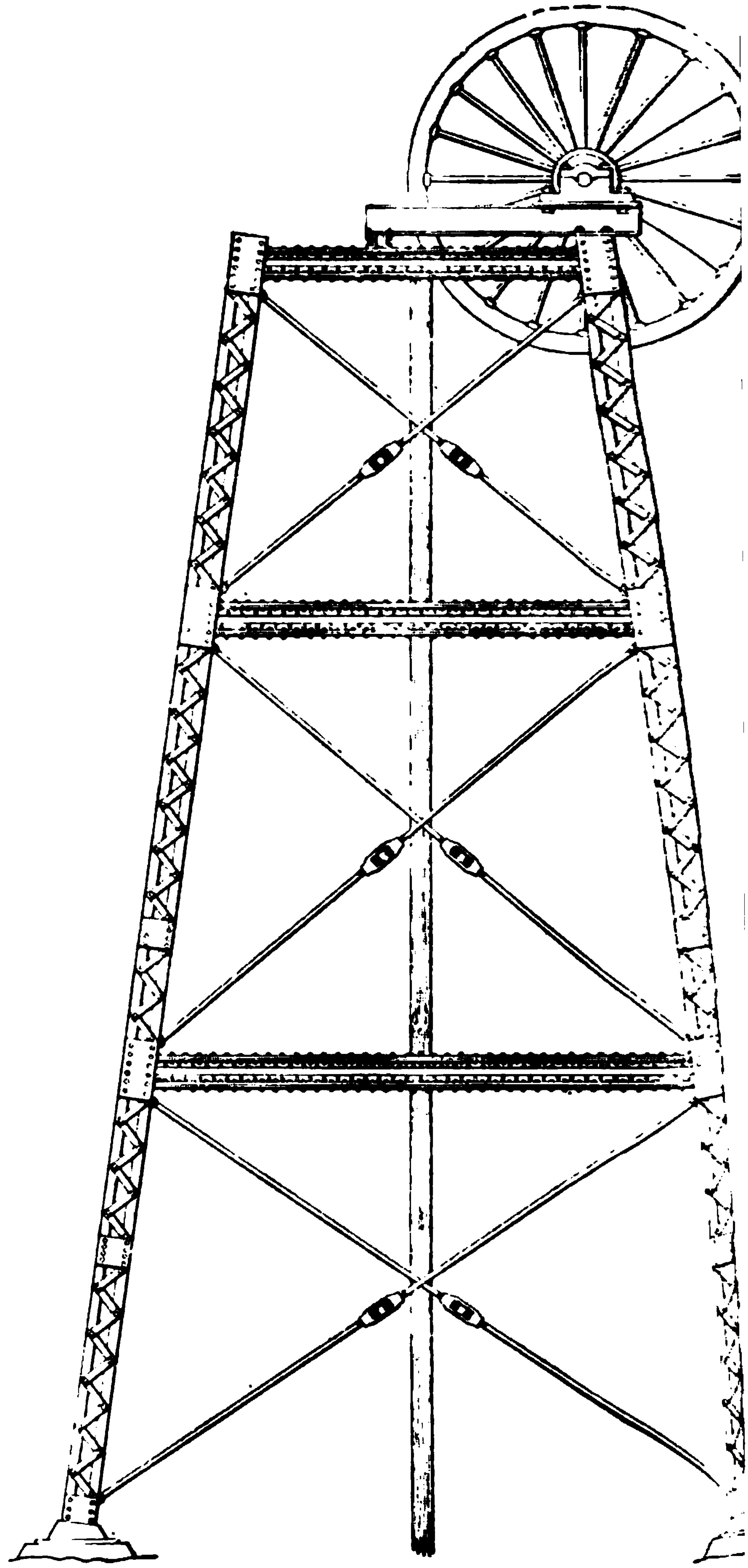
FIG. 944.

2565. So far, we have had in mind head-frames that were to be built of wood, but many of them are now built of iron or steel. These may follow the same lines as laid down in the foregoing pages, but they may also follow another line. Suppose, for instance, that we have a case such as is shown in Fig. 945, where, for some local reason, we

find it impossible to put a strut in or near the line of the resultant stress on the head-frame. This stress, then, has a component which tends to turn the head-frame over, and if it had only its weight to keep it in place, that might be the

Fig. 945.

result. Now, in iron or steel structures, we can very easily make A a tension member and anchor the lower end of it to a heavy masonry foundation. This resists the tendency to overturn and makes a very stable structure. A head-frame similar to this is shown more in detail in Fig. 946.



FIG

SURFACE ARRANGEMENTS
OF
BITUMINOUS MINES.

CHARACTER AND EXTENT OF SURFACE ARRANGEMENTS.

2566. The arrangement of the surface works of a bituminous coal-mine should be considered with regard to facilitating the operations inside as well as outside.

The rapid and economical handling of the coal when it is in readiness to be removed from the mine, the prompt returning of empty cars, and the supplying of the inside requirements of the mine should be kept constantly in view, since on the promptness and economy with which these demands are met will depend the capacity and profits of the operations.

The inside operations are directly dependent upon the efficiency and relation of the outside arrangements for the hoisting, ventilating, pumping, and possibly for the mechanical haulage, coal cutting, etc.

The relation of the surface works to the outside operations, as far as their general arrangement is concerned, is of special importance. On this will depend the efficiency in handling the coal and material to and from the mine, as well as the return of empty cars. This efficiency is secured by means of cars, landings, tracks, trestles, scales, tipples, chutes, screens, etc.

Although these are of immediate importance, they are dependent for their maintenance and continuance in operation upon other outside arrangements, as shops, sources of supply, preparation of material, facilities for shipments, etc.

2567. The arrangement of the surface works of a mine, as regards magnitude, design, and disposition, will depend upon the location, daily output, and nature of the operations, as well as the life thereof.

In some instances, the territory to be operated on is very small, with coal at shallow depths, and possibly subject to an increasing inflow of water with the breakage of the roof, so that the outside arrangements should be constructed with the view of moving them bodily to a new location with the least possible dismantling of buildings and structures. An example of this is furnished in the Braidwood region, Illinois, where, after mining the coal in 160 acres—by circular long-wall work, at depths of 100 ft.—the buildings, head-frame, etc., are moved on trucks to the center of another 160-acre tract.

2568. The design of the surface works will depend upon whether the coal is to be loaded on railroad-cars, boats, or dumped into bins for coke-ovens, or whether the coal is to be stored, on account of irregular shipment.

The method of mining, the amount of rock to be handled, and whether the coal is to be screened, picked, or washed, will also influence the surface arrangements.

The thickness of the seam and nature of its roof govern the use of large or small cars. These, with the arrangements for hoisting one or more cars at a time, influence the arrangement of the landings, their length, width, and the gauge of the tracks.

The nature of the labor employed or the possible output per miner, due to difficult mining, may make necessary such a large force of men, or such frequent changing of shifts, as to render it necessary to have an independent opening at a shaft mine for the lowering and hoisting of men, so as not to interfere with the hoisting of coal. In most regions a second opening is required by law.

If much rock is to be hoisted from the mine, or much timber and material is to be lowered, a second opening for the work may be necessary.

2569. The arrangement of the surface works will be influenced by the number and location of the mine openings. If the hoisting, ventilation, and pumping are to be performed through one opening divided into compartments, the arrangement of the plant around these openings should be carefully planned.

If there are two openings, the hoisting should be done at one and the ventilation at the other. The pumping may be done through either opening, although in the case of a rod or Cornish pump being used, its location in a compartment of the hoisting shaft is preferable, unless there is an independent opening or bore-hole for pumping.

The location of the plant on the surface should be carefully considered with relation to existing or proposed openings for conducting underground the ropes for haulage, or compressed-air pipes, or electric wire for motors, coal cutting, or pumping.

A mine may be opened with many of the surface arrangements of a temporary character, with the view of more permanent improvements, provided the life of the mine will permit. This plan is preferable if it is impossible to determine what the future requirements of the mine will be.

It is important that the few permanent features of such a plant be carefully planned, for, when once made, they must stand. If they are poorly planned, their inefficiency can not be entirely remedied. Some dependent portions of the operations will generally necessitate the continuance of some parts of the old work.

2570. In order to better understand the features in the surface arrangements, which will vary according to the nature of the opening, they can be best considered under the conditions of mines opened as follows:

1. By a shaft.
2. By an opening at a lower level than the dumping platform, necessitating a slope hoist. This will generally be a slope mine, although the coal from a drift or other opening may have to be hoisted by a plane before finally

reaching the dumping platform and surrounding surface works.

3. By a drift opened on a level with the dumping platform or at a point from which cars can run by gravity thereto.

4. By an opening in the mountain-side where cars can be lowered to the tipple by a gravity plane.

In each of these cases the details of the arrangements may vary in a few important points, owing to different conditions.

These particular conditions will be considered under each of the heads referred to.

Surface arrangements will be described to meet all the various requirements. They can be modified for operations where the outside requirements are few and simple.

SURFACE ARRANGEMENTS AT A SHAFT MINE.

GENERAL PLANS.

2571. The principal points in which mines opened by shafts differ from mines opened otherwise are as follows:

1. In the requirement of a head-frame for the vertical hoisting and landing of the loads, generally in single cars, requiring continuous and rapid hoisting.

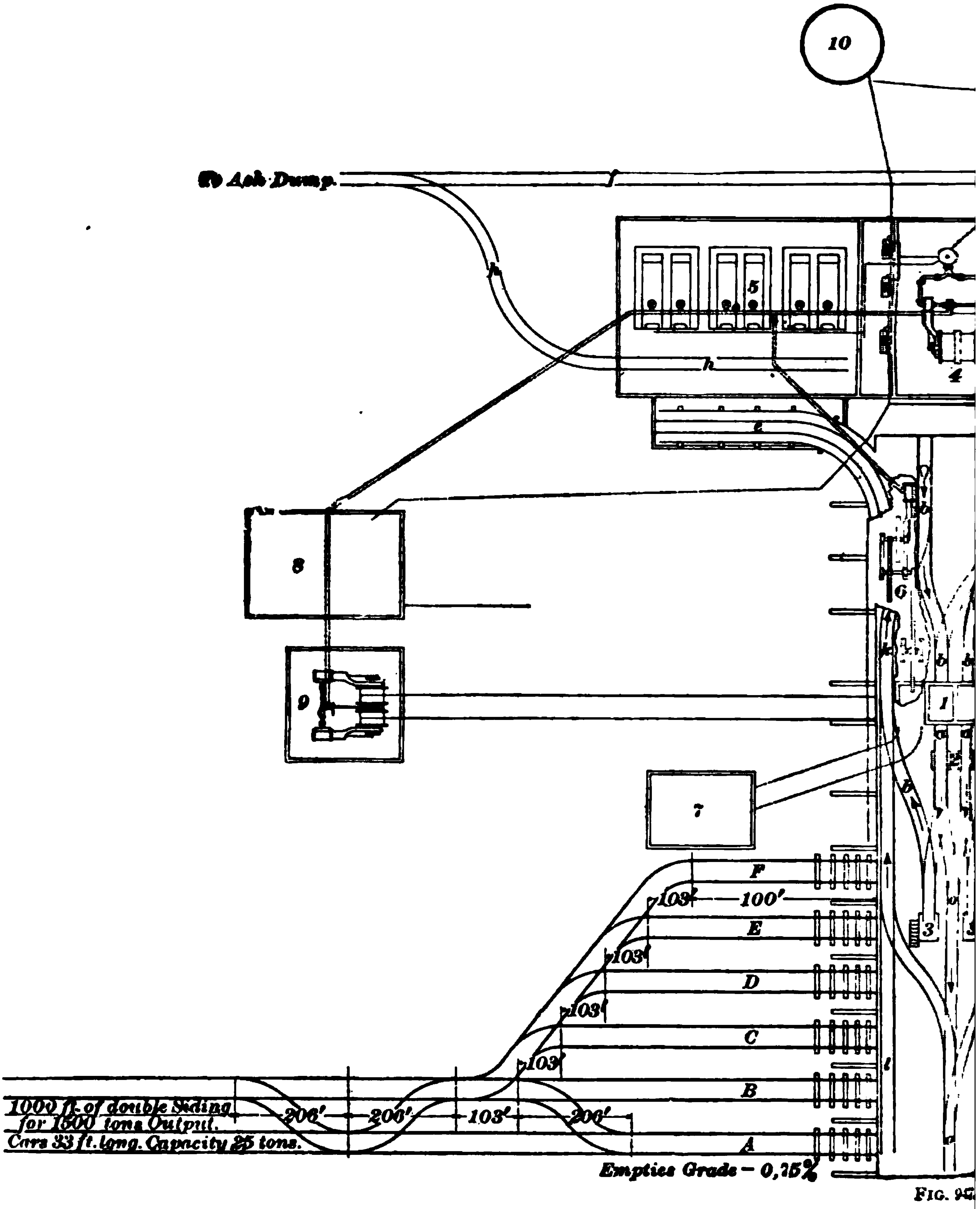
2. The location of the engines with reference to the head-frame and shaft, which is generally on line therewith and only a short distance away.

3. In the arrangement of the tracks at the landings, and of those leading to the dumping points and to the yard.

4. Provision for lowering and hoisting men and the handling of material up and down the shaft, especially large timbers, rails, and machinery, also mules and feed.

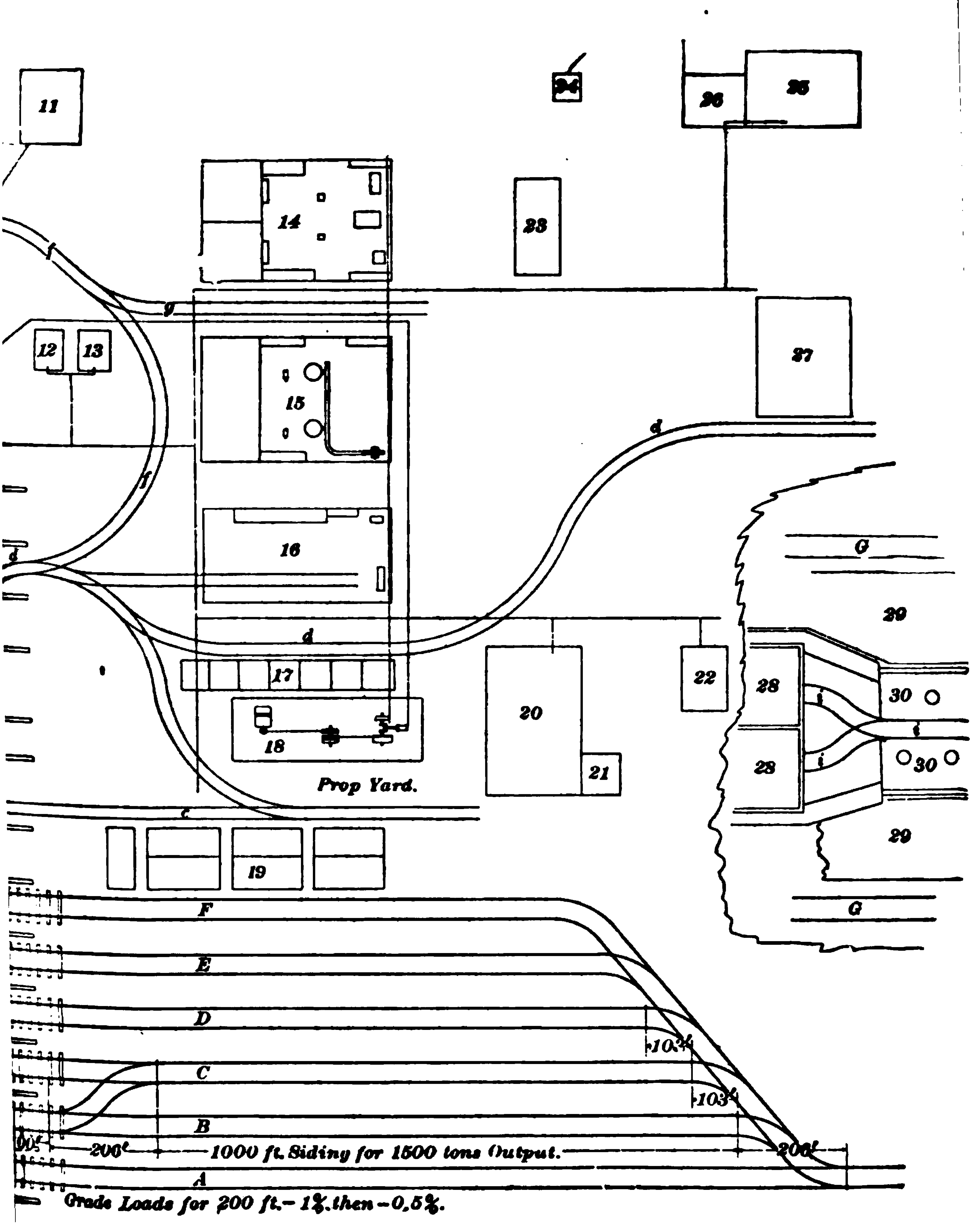
5. Arrangement of ventilating, pumping, or haulage plants near a compartment of the hoisting shaft, or at an isolated opening.

2572. There are two cases in the surface arrangements of shaft mines, as follows:



To Ash Dump.
10
5
4
8
9
7
6
1
3
F
E
D
C
B
A
103'
100'
206'
1000 ft. of double Siding
for 1500 tons Output.
Cars 33 ft. long. Capacity 25 tons.
Empties Grade – 0,25%

FIG. 9…



11
14
23
24
25
26
12
13
15
16
27
17
18
20
21
22
28
28
29
29
30
30
G
G
g
f
d
d
d
c
i
Prop Yard.
19
F
E
D
C
B
A
103'
103'
200'
200'
1000 ft. Siding for 1500 tons Output.
Grade Loads for 200 ft.–1%.then–0,5%.

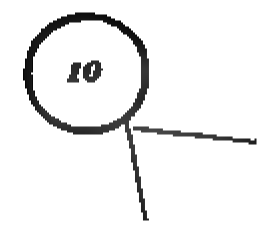
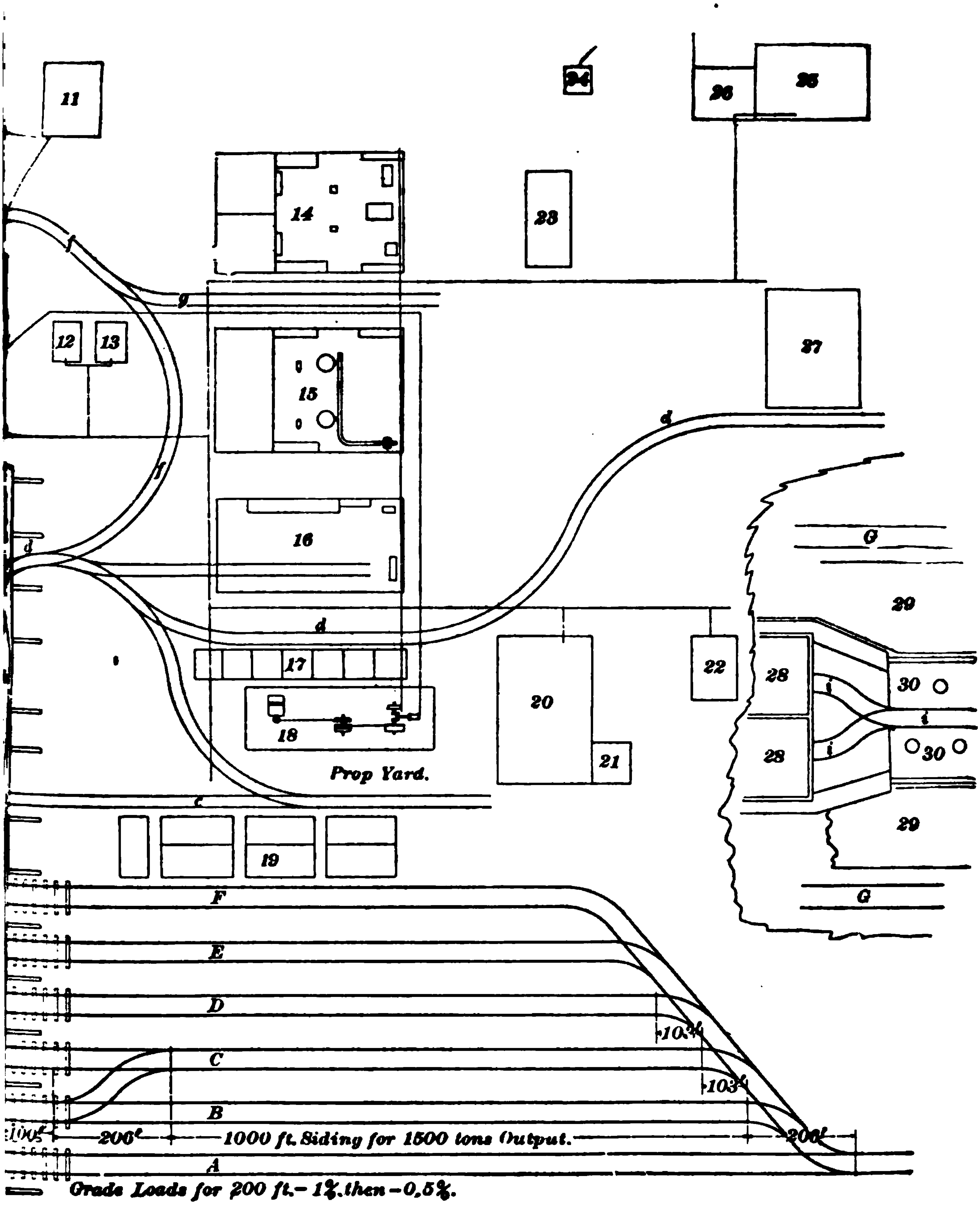


FIG. 92.



11
14
23
24
25
26
12
13
15
16
17
18
19
20
21
22
27
28
28
29
29
30
30
G
G
d
d
d
f
f
g
c
i
i
i
Prop Yard.
F
E
D
C
B
A
10%
10%
100'
200'
1000 ft. Siding for 1500 tons Output.
200'
Grade Loads for 200 ft.– 1%, then –0,5%.

Case I.—In this case there is one landing on a trestle leading to the tipple, and another at the yard level leading to the shops, timber-yard, etc. A general arrangement of this case is shown in Fig. 947.

This arrangement is the more common one for shaft mines, and is adopted where the ground in the neighborhood of the mine opening and the dumping point is level, or nearly so, and the two are not far apart.

2573. By referring to the plan, Fig. 947, it will be seen that the center lines of the hoisting-engine *4*, the shaft *1*, and the tipples *3* are the same. The boiler room *5* is located near the hoisting-engines, with a room between for feed-pumps and the fire-pump. The coal-bin for boiler supply is located in front of the boiler room. To the left of the shaft, and on a level with the surface, is shown the Cornish or bull pump *6*. The fan house *7*, shown on the left of the tipple, is intended for a fan to be used in case of accident to the regular fan at the air-shaft, which may be several hundred yards away. The dotted lines leading from the fan house to the shaft are intended to show the underground connection of the fan with the air compartment of the hoisting shaft. The two remaining buildings on the left of the tipple are the compressed-air or electrical plant *8* and the rope-haulage engine-house *9*. From the former the air-pipe or electrical conductors are taken into the mine through the air compartment of the shaft. From the latter the ropes run underground in wooden conduits to the shaft, and reach the workings through the air compartment of the shaft.

The tank for the water-supply *10* is shown back of the boilers, with service-pipes leading from it to the boiler feed-pumps, the compressor plant, the wash-house *11*, to all other buildings on the right of the tipple, and to the coke-ovens *30*. The mine office *12* and lamp house *13* are shown near the shaft and hoisting-engine house. The machine-shop *14*, blacksmith shop *15*, carpenter shop *16*, sawed-lumber yard *17*, saw shed *18*, and the timber and rail yard

19 are arranged almost in line and near the shaft. The machine-shop *14* has two rooms cut off from the main room, one for the storage of special fittings and the other for the head machinist's office. Besides the usual small tools, the machine-shop should be supplied with a punch machine, a drilling-machine, a lathe, a planer, and an emery-wheel. The necessary benches and closets should be built in where most convenient. The blacksmith-shop *15* should have a room on the side next the shaft for tools, and racks for picks needing sharpening and those sharpened. It should have two anvils, two forges, and a blower, and the usual complement of benches, closets, sledges, hammers, files, tongs, etc., etc.

The carpenter shop *16* should be so built that a track for crippled mine-cars can be run in one side of it. It should adjoin the blacksmith shop and be near the shaft, tipple, and sawed-lumber yard. Besides the work-bench, closet, bins for nails, etc., it should be supplied with a grindstone and a wood-turning lathe. The sawmill *18* and the sawed-lumber yard *17* should be located as convenient to the various shops as possible. In this case, the arrangement is all that could be desired, and the locations of the buildings are such that the problem of furnishing power from the sawmill engine to the machine tools in each shop is a simple one, and is solved by a straight line of shafting. The storage house for iron and pipe *23* is close to the machine and blacksmith shops, as it should be. The supply house *20*, supply-clerk's office *21*, and the oil house *22* are located in close proximity to each other, and near the shaft and shops. The stables *25*, harness and wagon house *26*, and the hay and feed storehouse *27* are another group of buildings that should be close together and convenient to the shaft. The powder house *24* should not be nearer any other building than 1,000 ft. The arrow attached to the building shown on plan indicates the approximate direction from other buildings for its location. The coal-bins for coke ovens *28*, the ovens themselves *30*, and the coke-wharves *29* are of necessity shown on the plan very near the other buildings. The bins *28* may be either at the tipple or at the head of the line of

ovens, about 300 ft. away from the shaft. They should be so constructed that the larries can run under the chutes and thence to the coke-ovens. The wharves *29* should be so arranged as to permit of the convenient drawing of coke, with some room for storage, and of such a height above the coke-car tracks as to permit of the easy loading of coke into the coke cars.

2574. The mine-car tracks on the surface are so arranged as to permit of the most convenient handling of both loaded and empty cars, to and from all necessary points on the surface. Tracks *a* are those on which the loaded cars run from the cage, over the scales *2* to the tipples *3*. The empty cars return to the cage by running around either side of the shaft on tracks *b*, and are back-switched onto the cages. All these tracks are arranged, by means of automatic hoists in the empty tracks at *k*, so that the cars are run automatically from the cages to the tipple, and thence back to the cages. Track *l*, on the platform in front of the tipple, is for the storage of cars of dirty coal until such a time as they can be picked over. Track *m* is for cars that need slight repairs, and which can be conveniently repaired without running them to the shops. Track *o* is for cars of rock, which are run across the railroad-tracks on the platform on the trestle, and are dumped in a convenient place. The arrangement of these tracks and the methods of handling the cars, in connection with the track *e* for coal to boilers, can be readily understood by referring to the plan.

The arrangement of the tracks on the surface is such as to enable timber, supplies, etc., to be taken into the mine speedily and conveniently. Track *c* running alongside the timber and rail yard *19* is used for conveying heavy timbers and rails to the shaft. Track *d*, which runs from the surface landing of the shaft to the hay and feed storehouse *27*, is for conveying sawed lumber from the lumber-yard *17* and feed from the feed storehouse *27* to the shaft. A short piece of track connects track *c* with track *d*, so that mine-cars can be loaded with props, ties, laggings, etc., and

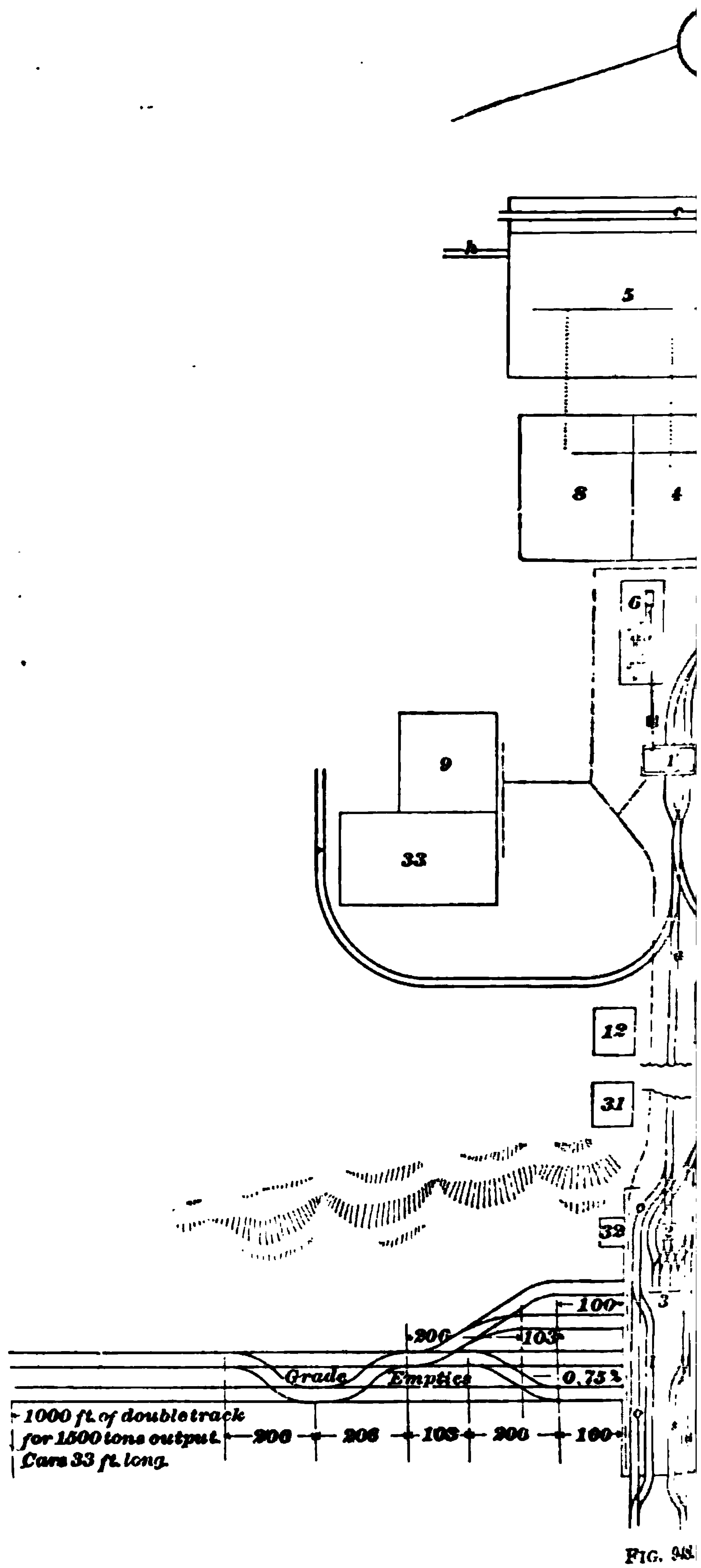
be run onto the cage at the surface landing. There is also a short branch track run off from track *d*, which enters the carpenter shop *16*. This track is used to run crippled cars to the shop and new cars from the shop to the shaft. Track *h*, which runs from the front of the boilers *5* to the ash dump, is used exclusively for the removal of ashes. Track *f*, which runs from a point on track *h* back of the boilers *5* and hoisting-engine *4* and connects with track *d*, is used to take ashes into the mine for road ballast, etc. A short branch track *g* is run off this track to the machine-shop, so that heavy pieces of machinery destined for use inside may be conveyed to the shaft on trucks, or if used outside they can be conveyed in the same manner to almost any part of the plant. The water-main from the source of supply may enter the tank *10* from any direction. The water-supply pipes from the tank to various parts of the works are shown by broken lines, thus ————. The steam-pipes from the boilers are shown by dotted lines, thus ....

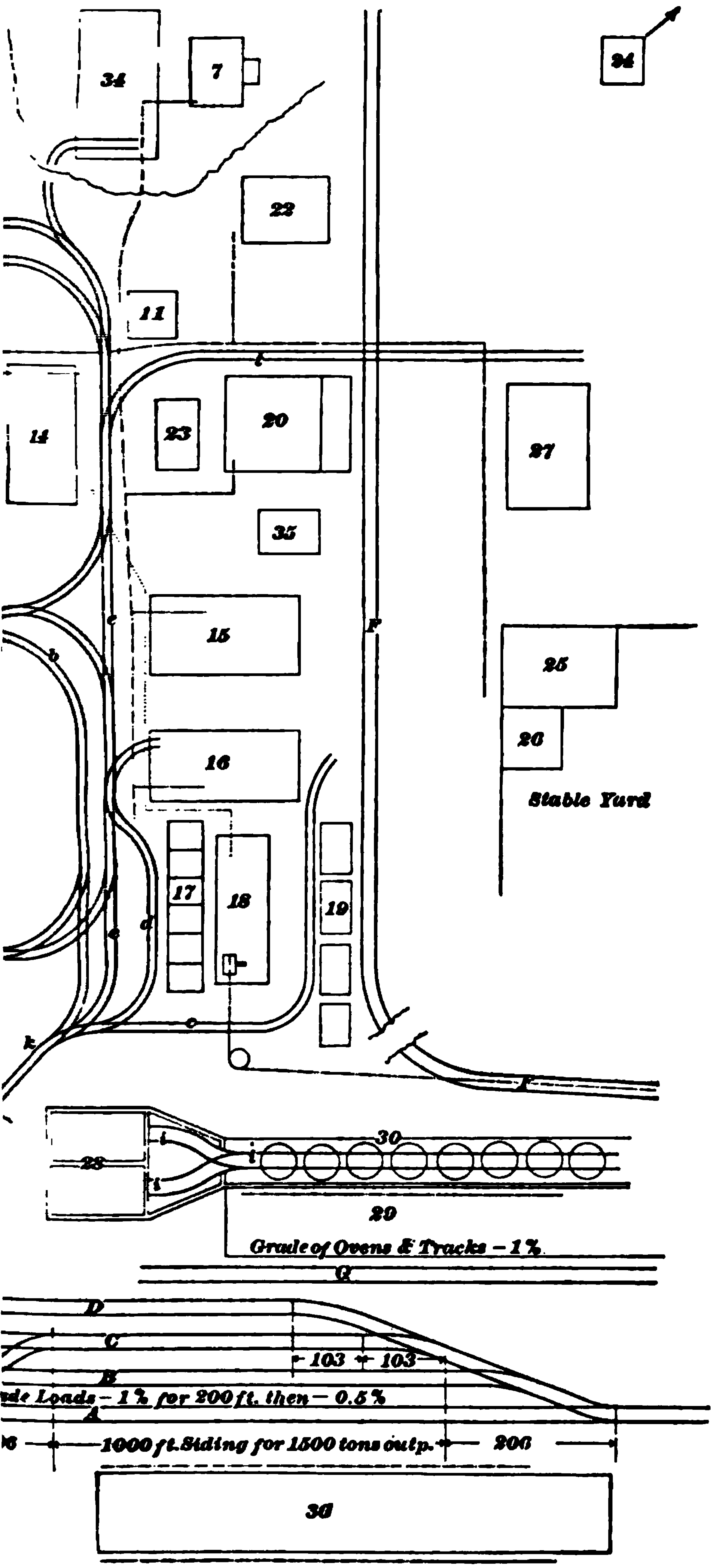
In studying this plan, the student should remember that it is for a large colliery, and that it is an ideal plan, and, therefore, is subject to many modifications. It is given simply as a general guide in laying out the surface plant.

If the coal mined is used for coke-ovens, the larry track from the coal-bins to the oven should be arranged as shown at *i*.

2575. The railroad-tracks for shipping purposes are shown on plan as follows: *A* is the empty shifting track, *B* the lump-coal track, *C* the nut-coal track, *D* the pea-coal track, *E* the slack-coal track, *F* the material track, on which supplies and machinery are received, and *G* the coke-car tracks. The curves at the switches in the shipping tracks are necessarily distorted. They should be 7° 30′ curves, and the frogs used should be what are designated as No. 8 frogs. But these are points that are decided by the engineers of the railroad company.

This arrangement is preferable to that in Case II, for the following reasons:


5
8
4
6
1
9
33
12
31
32
3
100
200
103
Grade
Empties
0.75%
1000 ft. of double track
for 1500 tons output.
Cars 33 ft. long.
200
206
103
200
100
FIG. 93



34
7
24
22
11
14
20
23
27
35
15
25
16
26
Stable Yard
17
18
19
28
30
29
Grade of Ovens & Tracks – 1%.
G
D
C
103
103
B
A
1000 ft. Siding for 1500 tons outp.
200
36

First.—The dumping point being nearer the other surface works, the operations are more concentrated, and, therefore, more readily supervised, and they are nearer the center of repairs and supplies.

Second.—The location of the storehouses and material and timber yards can be arranged conveniently to the railroad-tracks as well as to the mine, so as to require very little handling of material.

2576. **Case II.**—In this case there is only one landing, and that on the natural ground, the dumping point being some distance from the shaft.

The arrangement of the surface works and the tracks leading thereto will vary slightly, due to their having the same shaft landing as the cars to and from the mine. A suitable arrangement is shown in Fig. 948.

This arrangement may be made necessary by certain conditions in mining requiring a shaft location at a higher point than the level of the tipple dump, and sufficiently high for the necessary elevation of the tipple above railroad-cars.

2577. The principal features in this arrangement, as compared with Case I, are:

1. The head-frame is not so high, which is an advantage.
2. The engine will be nearer the shaft, which may be a disadvantage if more room around the shaft is needed.
3. The arrangement with one landing facilitates the handling of empty cars, material, etc., between the yard and the mine, although the arrangement of tracks may involve more turnouts, crossings, etc.
4. If supplies, lumber, machinery, etc., are received in large quantities by railroad, they will require considerable handling to bring them to the yard level. This may be effected by a wagon road, or, if the circumstances warrant, by an inclined track, up which the railroad-cars can be hoisted by a rope, as shown in Fig. 948, at *F*.

For this purpose the best plan of locating and grading an inclined track should be studied, in order that, in the

arrangement of the surface works, means can be provided for hoisting cars of material from the railroad-tracks below. Or, if railroad-cars of material are not to be hoisted, then a track can be laid for hoisting smaller cars which will come close enough to the railroad-track to transfer the material to be hoisted.

2578. Great importance is attached to the proper design, arrangement, and construction of the head-frame, for when once erected and the related structures have assumed permanent form, its alteration is a serious matter, especially if it involves a change in the surroundings. Therefore, the many points bearing thereon will be dwelt on in some detail, as will also the arrangement of the landings, trestles, tracks, and how the coal is to be handled at the tipple. The arrangement of chutes, screens, and whether the coal is to be picked, washed, or stored in bins, should be fully considered, so that the main outlines may be correct, and the proper height for dumping and sufficient distance between the shaft and railroad-tracks, or dumping points, may be secured.

2579. By referring to Fig. 948, it will be seen that the same general plan of concentrating the plant has been followed as in Fig. 947, but the locations of the improvements are somewhat different. In this figure, the shaft is shown at *1*, the scales at *2*, the tipples at *3*, the hoisting-engine at *4*, the main boilers at *5*, the Cornish or bull pump at *6*, the fan (in this case an isolated ventilating plant) at *7*, the boilers for the isolated fan at *34*, the air-compressor, or electric plant, at *8*, the underground haulage engines at *9*, and if the haulage plant is located away from the main shaft and the ropes conveyed into the mine through another shaft, or through bore holes, a boiler plant *33* is erected close by. The tank is shown at *10*, the wash-house at *11*, the mine office at *12*, the lamp house at *13*, the machine-shop at *14*, the blacksmith shop at *15*, the carpenter shop at *16*, the sawed-lumber yard at *17*, the sawmill at *18*, the timber and rail yard at *19*, the supply house, with supply-clerk's

office, at *20*, the oil house at *22*, the iron and pipe shed at *23*, and the isolated powder house (in general direction) at *24*. The stable *25*, harness and wagon shed *26*, and the hay and feed storehouse *27* are shown in close proximity to each other. The coal-bins for coking coal are shown at *28*, the coke wharf at *29*, and the line of coke-ovens at *30*. The check-clerk's office is shown at *31* and the weighman's office at *32*.

The mine-car tracks are arranged, as in Fig. 947, so as to make it possible to reach all parts of the plant with cars or trucks. Track *a* is the loaded track, passing over the scales *2* to the tipple *3*. Track *b* is the empty track, over which the empty cars run by gravity from the tipple to the mechanical hoist at *k*. Here they are raised so as to run by gravity to hoist *k'*, and thence they run to the shaft. The rock cars from the shaft are run to the rock dump over track *o*, and the cars of dirty coal are run on the track *l*. The heavy timber and rails are taken to the shaft from the yard *19* over the track *c*, and the sawed lumber is taken from the yard *17* to the shaft, tipple, or shops over track *d*. The cars to the shops and the coal to the main boilers are run over track *e*. The ashes from the main boilers are run over track *h* to the dump, or over track *f* to the shaft. Feed and hay are taken to the shaft from the hay and feed storehouse *27* over track *t*. This same track answers for supplies from the supply house *20*. Machinery taken to and from the machine-shop is hauled over track *e*. Track *u* on the tipple platform is for empty rock cars, and track *s* is for cars needing slight repairs. Coal for the isolated steam plants at the fan and haulage engines is transported over tracks running off from the main system at the most accessible points. The larry tracks to the coke-ovens are shown at *i*.

The coal-bins for coke-oven supply may be connected with the main tipple, or they may be some distance away, but with proper arrangement of height to run the mine-cars conveniently to the dumps located therein.

The shipping tracks are as follows: *A*, empty shifting

track; *B*, lump-coal track; *C*, nut-coal track; *D*, pea-coal track, and *G*, coke-car track. Coke-ovens may also be located on the opposite side of the railroad-tracks from the tipple, as at *36*.

The water-supply pipes from the tank are shown thus ——·——·——·——. The steam-pipes thus - - - - - - - -, and the rope for the inclined plane, in the material track *F* (worked by the sawmill engine), is shown thus ——· ·——· · ——· ·——.

THE HEAD-FRAME.

2580. The head-frame is for the support of the head sheaves, or wheels, over which the hoisting ropes are led from the hoisting-engine, located at some distance from the shaft, and for lowering, raising, and landing the cages.

Fig. 949 shows a general outline of a head-frame built of timber, and Fig. 950 shows a general form for one built of iron or steel.

The head-frame consists of an upright part sufficiently strong to support the load to be hoisted, whose weight is transmitted to the structure through the rope to the head wheel *C*, which may have a bearing directly over the upright posts, or the bearing may be on a stringer, or girder, spanning the shaft-way, and resting on posts on both sides of the shaft.

The posts *B* may be inclined in the direction of the length of the shaft, if necessary, to impart greater stability to the structure, and for its perfect bracing, especially if the head-frame is high.

They may also be inclined in the other direction, but this is not necessary unless it is desired to carry them to foundations somewhat distant from the shaft.

The tendency of the upright part of the head-frame to overturning when hoisting, due to the resultant of the forces acting thereon, is resisted by a strut, or brace, *D*, extending from the framework near the sheave to the ground, and in such a direction that the resultant of the forces will lie between the foundations of the upright and the brace.

This resultant force is due to the weight of the load and the force exerted by the engine in raising it, both equal, and exerted in the direction of the rope.

2581. Fig. 949 shows a timber head-frame in plan, side elevation and end elevation. A fan is shown at *E*. The yard landing is shown at *F* and the trestle landing at *G*. A self-dumping cage is shown at *H*, with a car ready for dumping in the chutes. The levers for opening the chute gates are shown at *I*. The bars for screening the coal, when dumped, are shown in plan at *K*. The shed *L* shown in side elevation is for a roof to the chutes, and for such tracks or machinery as may be required at the trestle landing.

2582. In Fig. 950, which is a side and end elevation of an iron or steel head-frame, the yard landing is shown at *F*

FIG. 950

and the coal landing is shown at *G*. The location of a pump is shown at *E*.

2583. The direction and intensity of the resultant of the forces acting on the head-frame are determined by means of the parallelogram of forces.

In Fig. 951, *E C* represents the direction of the rope from

the engine to the sheave, and *D C* the direction of the rope from the shaft to the sheave. Their intersection is at *C*.

If the weight of the cage, car, load, and rope is 8,000 lb., and if a scale of 4,000 lb. to the inch is assumed, a distance can be laid off from *C* to *D*, 2 inches long, which will represent the direction and intensity of strain due to a vertical lifting of the load. The weight of the cage, car, load, and rope will produce the same strain in the rope in the direction *C E* that it does in the direction *C D*, so that from *C*

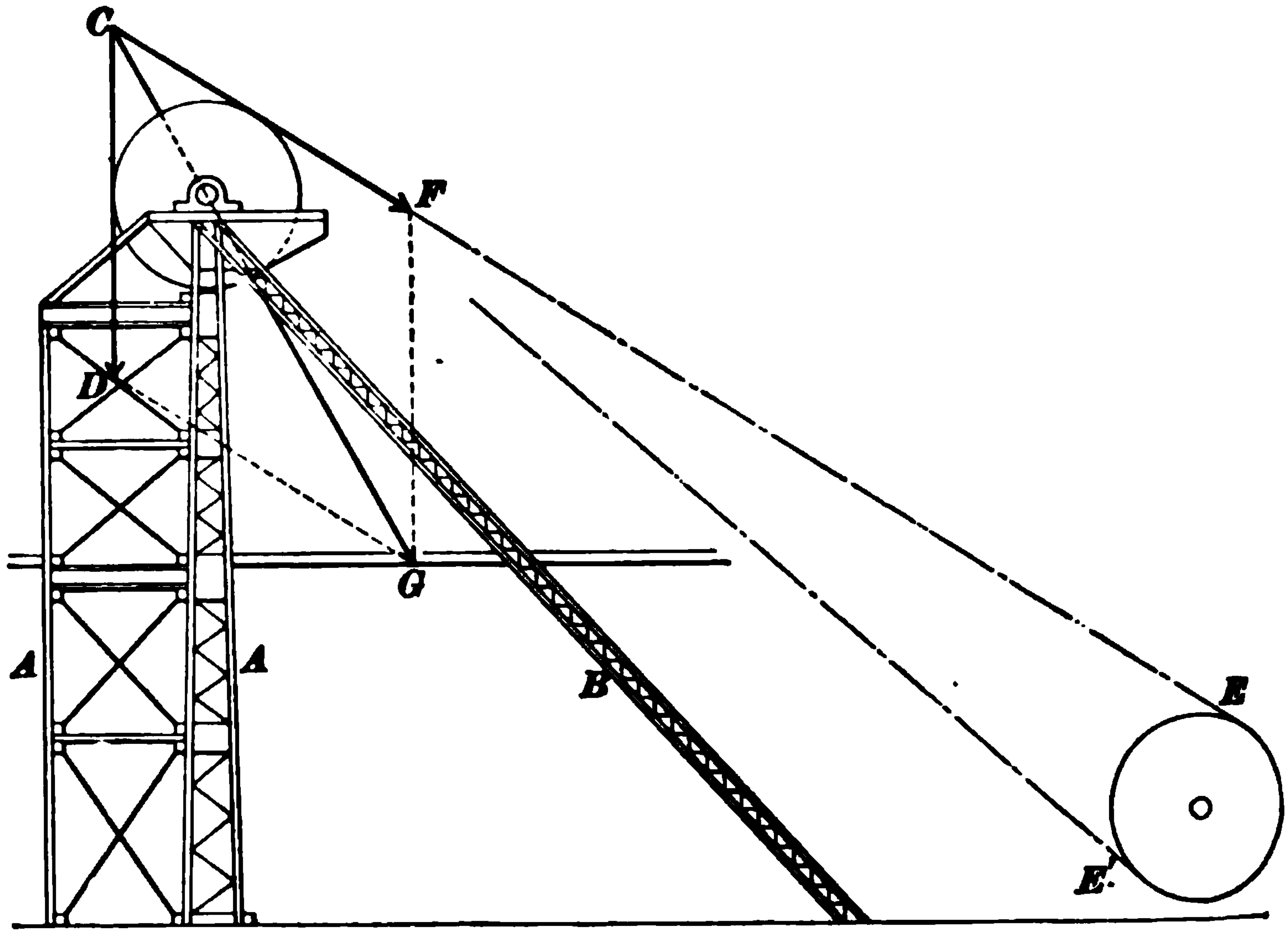


FIG. 951.

a distance *C F* can be laid off representing a strain of 8,000 lb. On a scale of 4,000 lb. to the inch, *C F* will be laid off 2 inches long.

The resultant is now found by drawing a line *F G* parallel with *C D*, and *D G* parallel with *C F*. From the intersection *G* of these lines, a line is drawn to *C*. Then, *C G* will represent the resultant in direction and intensity which, measured to the scale, will show a strain of between 14,000 and 15,000 lb. The direction and intensity of the resultant will vary if the direction of the underwinding rope is also considered.

The strut, or brace, B is designed to resist this thrust, and where there are no other strains to be considered the direction of the brace would be theoretically in the direction of the line $C G$. There are vibrations caused by the variable strains in hoisting, and by the movement of the ropes, which would subject the structure to injurious strain if the braces were too near the line of the resultant.

Therefore, the direction of the brace will be somewhere between the resultant and the line of the underwinding rope of the engine, or its direction will be such that the angle between it and the resultant will include $\frac{1}{3}$ or $\frac{1}{2}$ of the angle formed by the resultant and the line of the underwinding rope.

The brace should have a batter of from 1 to 2 inches to the foot, so as to be wider at its foundation than at the top, for greater stability and better bracing.

It may be found necessary to provide space for a Cornish or other pump near the shaft, or for the movement of cars around the shaft. This may be arranged by drawing in or spreading the brace, provided the limits indicated are not exceeded.

2584. In the design of the head-frame, proper clearance in the structure should be allowed for the movement of the cars at the landings, and for passing cars around to the rear of the shaft, if such an arrangement is desired.

The material of which the head-frame is built will be wood, iron, or steel. Generally the former is selected for moderate outlay, or where operations are of a more temporary character. Iron or steel head-frames are adopted for operations of greater magnitude, for high head-frames, or where timber is scarce, or where hoisting is deep and rapid. The strains are better resisted in a metal structure than in a wooden one.

Wooden head-frames are built with each post of single timbers 12, 14, or 16 inches square, or they may have composite posts made by joining and bolting 6″ × 12″ or 7″ × 14″ timbers. In this case, the uprights supporting the sheaves

have 4 such timbers in each post. The upright on the opposite side of the cage has 2 such timbers in its posts, as has also the main brace.

Iron or steel head-frames are built of angle-irons, with web of lattice or plate. Channel-iron is also used, joined with lattice bracing, forming a hollow post.

Angle-irons are generally $2'' \times 2\frac{1}{2}''$, $3'' \times 3''$, or $4'' \times 4''$, used in sets of 2 or 4, braced with $\frac{1}{2}'' \times 2''$ iron, about 8 inches deep or more. Channel-irons are generally 8 to 12 inches deep, joined with $\frac{1}{2}'' \times 2''$ braces.

The height of the head-frame to the sheave center varies from 35 to 50 ft. at shafts with only one landing, and from 50 to 85 ft. in height for shafts landing on a trestle. The distance from the foundations to the tipple landing depends upon the height above railroad-tracks needed for dumping or for filling bins, supplying coke-ovens, etc.

2585. Where the coal is to be loaded on railroad-cars, and only one or two sizes are produced without much handling or screening, a height of 24 feet is sufficient; for producing three or four sizes, a height of about 30 feet is required; and if any handling or cleaning of the coal is necessary, a height of 35 or more feet is necessary, unless the ordinary bar screens are replaced by a shaking screen and elevators to diminish the height of the structure. Where coke-ovens are to be supplied, a height of at least 30 to 36 feet above the railroad-tracks at the coke-ovens is required for dumping directly into the larry without the intervention of bins for storing coal. Bins should be provided, and generally a height of 60 to 65 feet for the tipple platform above railroad-tracks at coke-ovens will afford sufficient bin room.

This height of the tipple platform may be reduced if screenings are to be used in coking, or the coal is to be washed. This will necessitate some intermediate handling of the coal by conveyers or elevators, in passing it from the tipple to the coal washer or coke-ovens. In this case it is preferable that the height of the structure be a minimum.

The screenings may be raised by elevators or in cars on a plane to the dumping points at the ovens or the washer.

The height from the tipple platform to the center of the head sheaves varies from 25 to 40 feet, depending upon the height of the cage and the clearance needed for safety devices. A height of 35 feet is generally sufficient.

DISTANCE OF ENGINE FROM THE SHAFT.

2586. The engine should be located at a sufficient distance from the shaft to allow the rope to wind evenly on the drums, and so that the coils on the drums will not loosen when the cages rest on the landings. The longer the lead of the ropes, the better they will coil on the drums.

In some cases the location of the engine near the hoisting shaft, with the ropes leading therefrom in a nearly vertical position, can not be avoided.

Too great a distance of the engine from the hoisting shaft, with heavy ropes, may result in the weight of the rope between the head sheaves and the engine raising the cage a few inches from the tipple landing when the loaded car is removed.

It is usual to make the horizontal distance of the center of the engine-drums from the center of the shaft from one to one and a half times the vertical height of the center of the sheaves above the center of the engine drums.

For high head-frames this horizontal distance is 1 to 1.3 times the vertical distance; for low head-frames from 1 to 1.5 times the distance.

If in a head-frame this vertical distance is 70 feet, then the horizontal distance will be from 70 to 91 feet. If the height is 50 feet, the horizontal distance will be 65 to 75 feet.

2587. The end of the engine foundation is sometimes arranged for the foundation of the back brace of the head-frame. In this case the brace is caused to lie at a considerable distance from the resultant. The arrangement of the tracks at both landings and the possible future need of structures near the shaft for pumping, ventilation, etc.,

should be considered in connection with the planning of the head-frame. In order to provide the necessary clearance and space, the posts of the upright part of the head-frame can be planned near or at some feet from the shaft, and with little or considerable batter, provided the limits are not exceeded.

The points where the posts and the brace penetrate the platform and strike the ground should be noted with relation to the proposed tracks. Either the arrangement of the tracks or the head-frame can be altered to suit, and the spread of the brace can be varied somewhat to provide some clearance in front of its posts or between them, at either the tipple or yard landing.

ARRANGEMENT OF TRACKS.

ARRANGEMENT WHERE THE TIPPLE LANDING IS ON A TRESTLE.

2588. The tracks between the shaft and tipple may be very short or long, varying with the particular conditions and requirements of the mine. There are three cases under which the length of tracks may be considered, viz.:

1. In the case of a self-dumping cage, as shown in Fig. 949, where there will be no tracks between the shaft and tipple.

2. Where the car is moved off of the cage a distance about equal to its length and dumped and returned, or, if the arrangement will permit, the cars are dumped on both sides of the cage, as shown in Fig. 952, in which *A* is the shaft and *B* the tipples on both sides of the shaft.

3. Where arrangements make it desirable to remove cars from the cage, and arrange passing tracks for loads and empties and for running rock cars to the dumps, which may be either in the coal tipple or at a distance. This third arrangement is in most general use.

2589. The arrangement of the self-dumping cage can be used, provided it is not desirable for any reason to frequently remove the cars from the cage at the top of the

shaft. Under certain conditions, the depth and rapidity of the hoist, size of load, the amount and handling of rock, cleaning of coal, and the steadiness with which cars can continue in use without removal from the cage outside, will determine its use.

Rock can be handled by a self-dumping cage by providing a rock chute at the rear of the shaft, or through a trap-door in the coal chute, as shown at *A* in Fig. 949.

If the coal requires picking to remove slate, it can be done by introducing a belt conveyor at the foot of the screens for

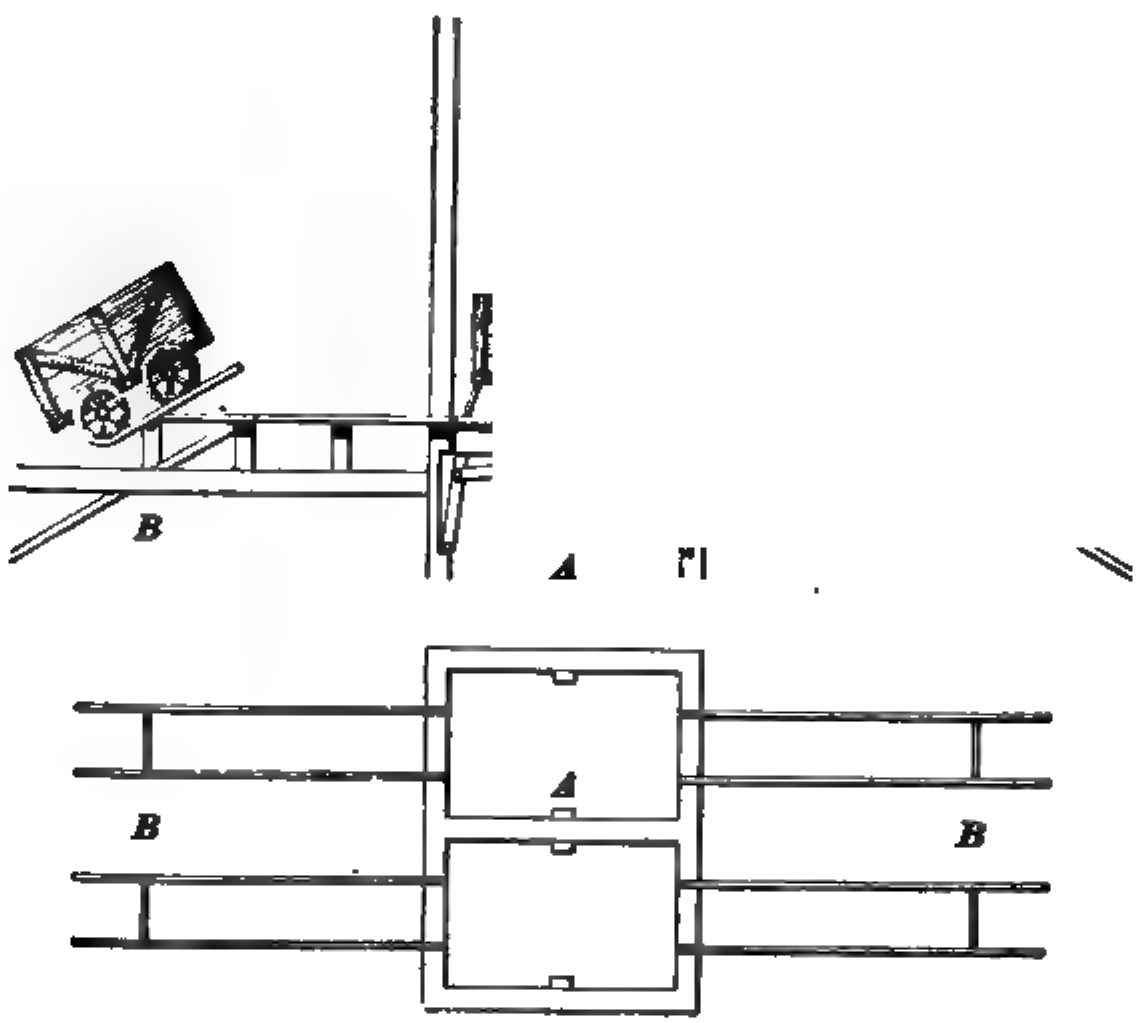


FIG. 952.

the lump coal to land on. This should be about 4 feet wide and long enough to allow sufficient time for picking before it discharges into a bin or the railroad-cars.

It is important that a self-dumping cage allows the coal or rock to free itself entirely from the car in dumping,

whether wet or dry, and that the time required produces no serious delay.

For the arrangement in the second case it is preferable that the coal be clean, and that little rock is to be handled, although, as in the first case, the rock can be handled and the coal cleaned as indicated. This plan also requires that cars must not be frequently changed outside.

2590. In the third case there are two arrangements, as follows:

1. Where both the removal of loaded cars from the cage and the return of empty cars thereto is done at the front of the shaft.

2. Where the loaded cars are removed at the front of the shaft and the empty cars are returned to the cages at the rear of the shaft.

In either case there will be one or more empty cars on the platform, in readiness to be put on the cages when the loaded cars are removed.

The distance needed between the shaft and the tipple will depend upon the arrangement of the tracks and the switching room required for the loaded and empty cars, and possibly for rock cars. Generally 20 to 60 ft. will be sufficient, although longer tracks can be used if necessary.

2591. First Arrangement.—In this case it may be necessary to dump the cars at one tipple. An arrangement

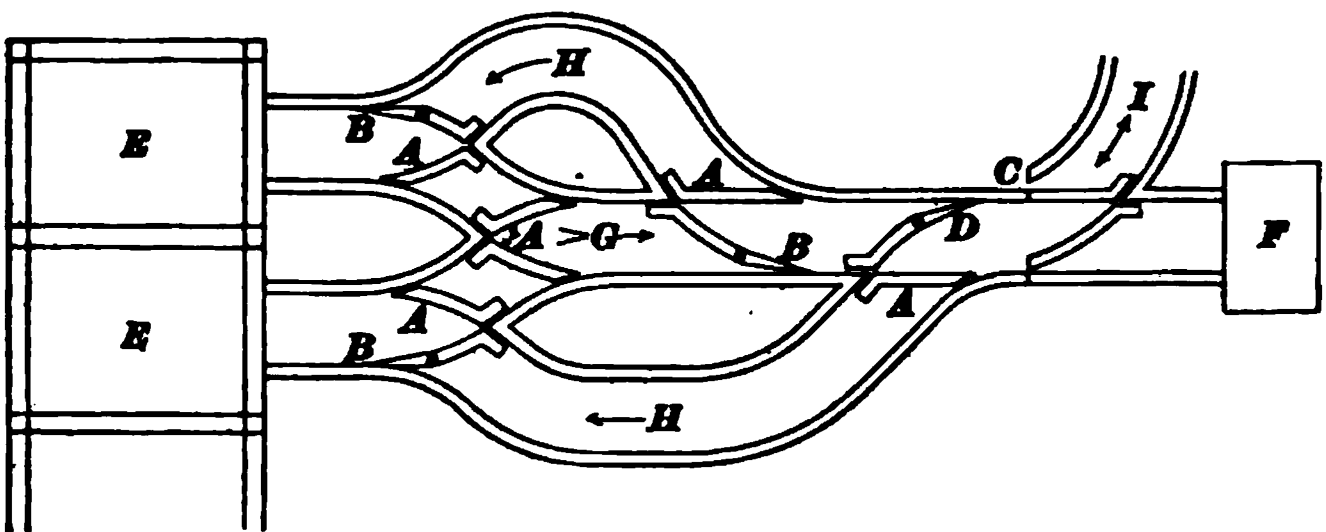


FIG. 953.

of tracks for this purpose is shown in Fig. 953, although where only one dump is employed it is preferable that it be one on the principle of the Mitchell, the Wilson, or the

Phillips dump, which allows the cars to pass beyond the tipple after dumping. These latter arrangements are described under the head of tipples.

2592. The arrangement shown in Fig. 953 requires longer tracks than where there are two tipples. In this arrangement the cageways are shown at *E*, *E*, the tipple at *F*, the loaded track at *G*, the empty tracks at *H*, *H*, and the rock track at *I*. At *A*, *A*, *A*, *A* are point-rails, at *B*, *B*, *B* spring-latches, at *C* a throw switch, or latches, and at *D* a latch which is moved by the foot.

Rock can be switched out as shown and carried to the rock dumps wherever located. An arrangement of tracks leading to two tipples is shown in Fig. 954.

In this case *E*, *E* show the cageways and *F*, *F* the tipples. The loaded cars run straight to the tipples and the empty

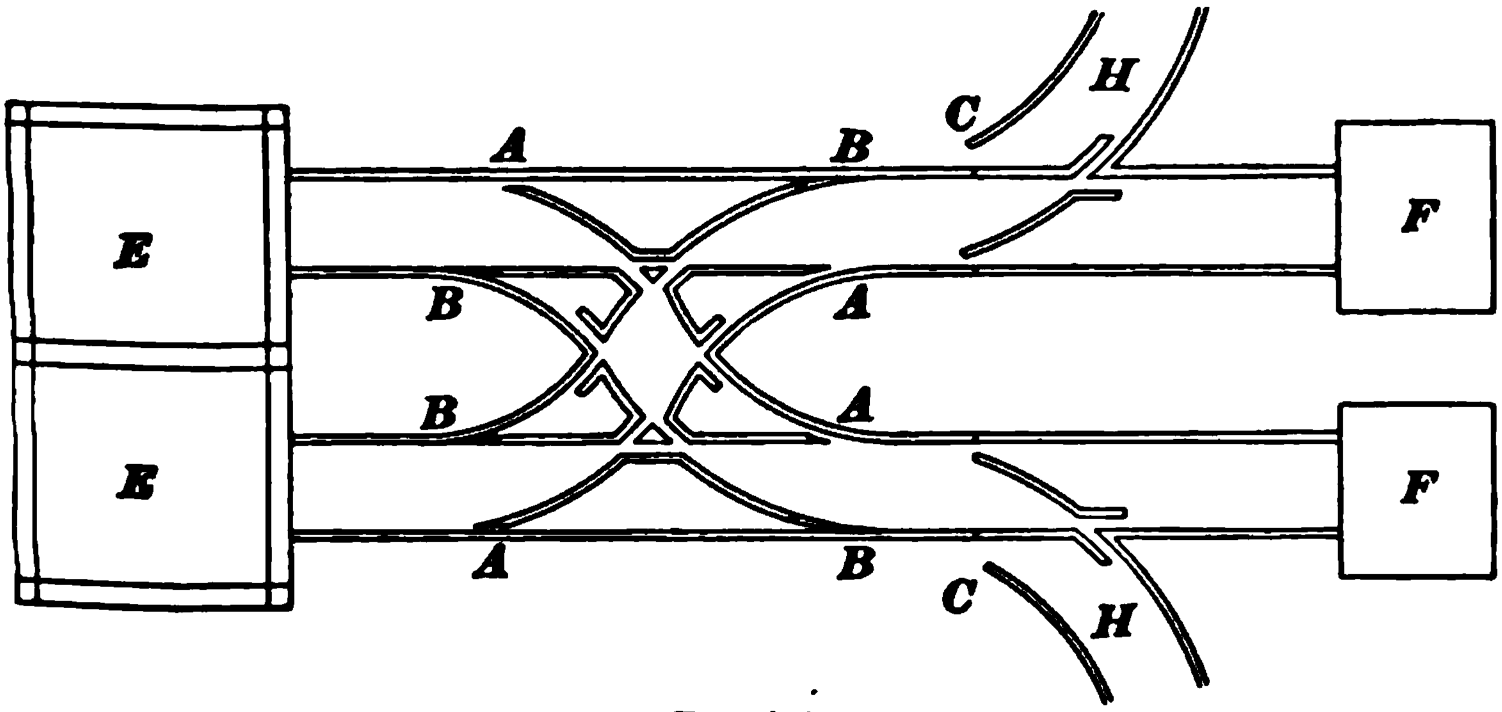


FIG. 954.

car is returned, by the curved tracks and switches, to the cage opposite the one on which it was hoisted and from which a loaded car has just been removed. The rock cars are run out on tracks *H*, *H*. By referring to the plan, it will be seen that an arrangement of point-rails at *A*, *A*, *A*, *A*, of spring-latches at *B*, *B*, *B*, *B*, and of throw switches at *C*, *C* will be very effective. These tracks require very little distance from the shaft to the tipple.

2593. Another arrangement, shown in Fig. 955, will permit of holding several empty cars. In this case,

however, the loaded and empty cars hoisted and lowered on the one side can not be crossed over to the other side, which is desirable in some cases.

2594. In Fig. 955, as in Fig. 954, *E*, *E* show the cage-ways, *F*, *F* the tipples, and *H*, *H* the rock roads. The loaded

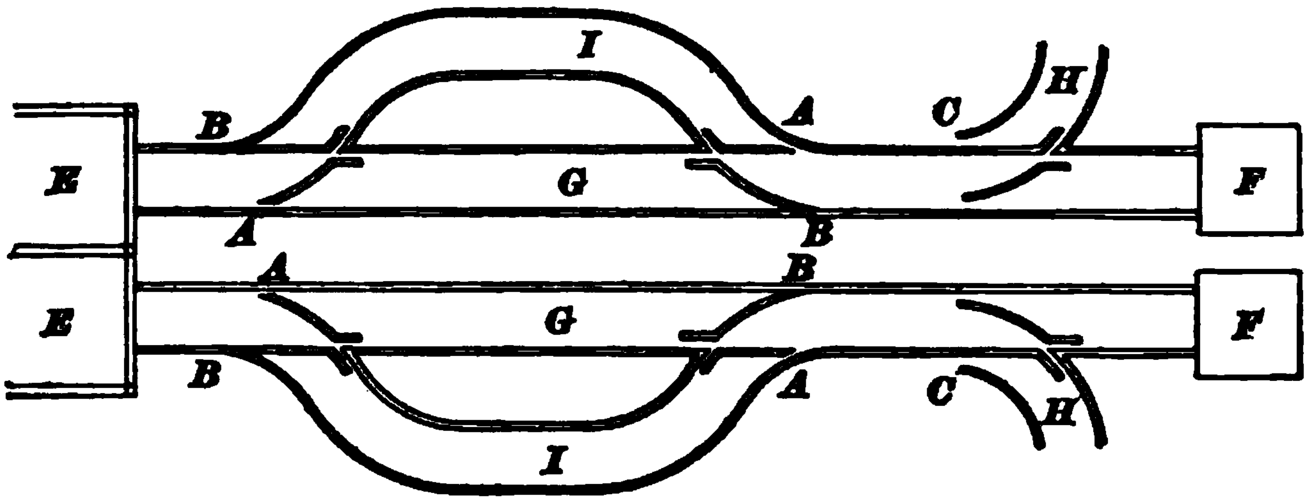


FIG. 955.

tracks *G*, *G* run straight from the shaft to the tipples, while the empty tracks *I*, *I* are of the nature of turnouts. The arrangement of switches is as follows: *A*, *A*, *A*, *A*, point-rails; *B*, *B*, *B*, *B*, spring-latches; *C*, *C*, throw switches, or latches.

If it is necessary to cross cars over from one side to the other, the two center tracks can be brought together, form-

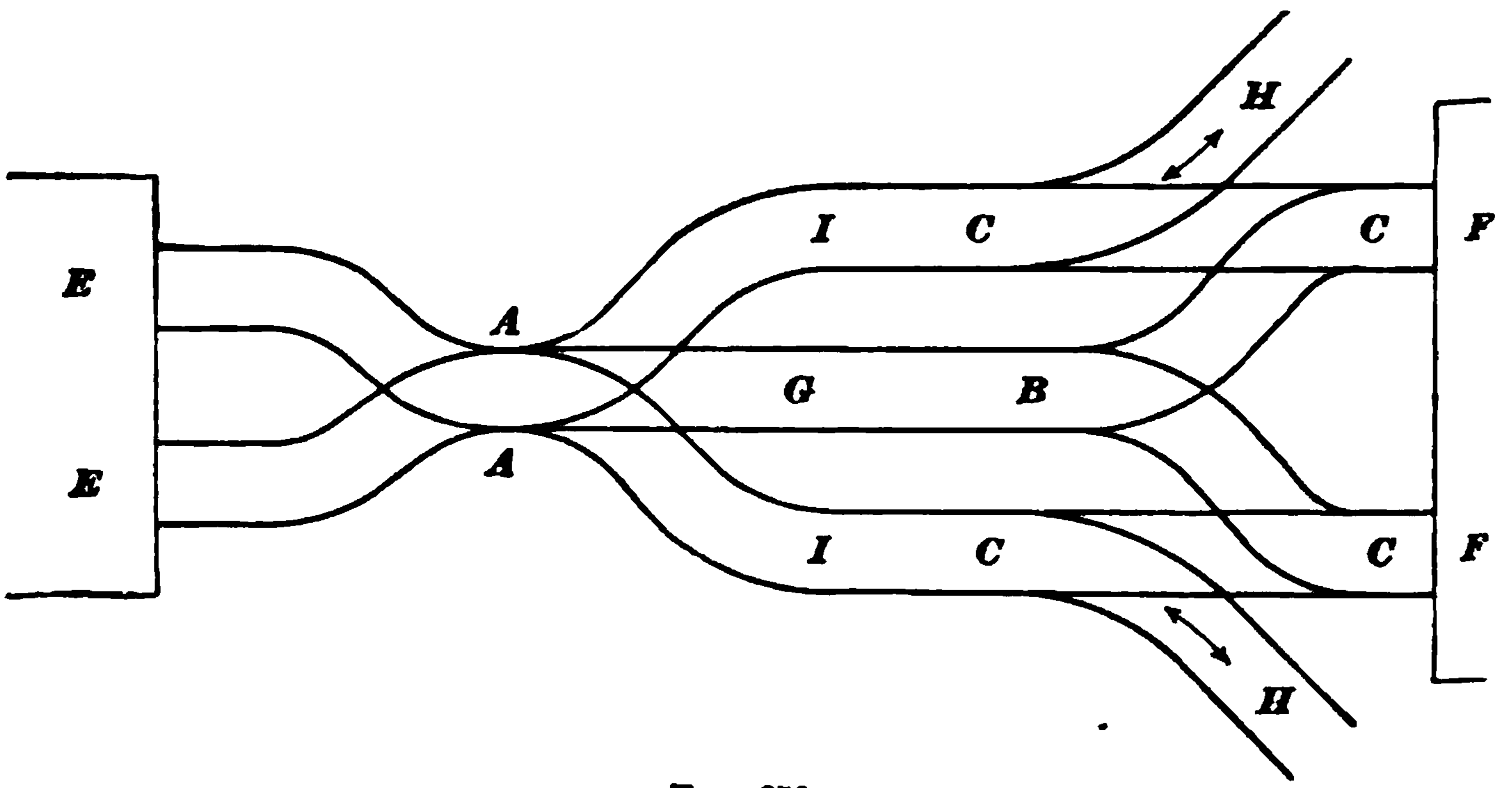


FIG. 956.

ing three tracks at the center, and resolving into two tracks at the shaft and tipple. This arrangement is shown in Fig. 956.

2595. In Fig. 956, *E*, *E* show the cageways and *F*, *F* the tipples. *G* is the loaded track, *I*, *I* the empty tracks, and *H*, *H* the rock tracks. *A* is a double throw switch, *B* spring-latches, and *C*, *C*, *C*, *C* single throw switches, or latches.

If there is any rock to be handled, it can be switched out, as shown, and led to distant rock dumps to the right or left or beyond the tipple, or to rock chutes in the tipple structure, where the rock can be held in bins, and finally removed in railroad-cars.

2596. Second Arrangement.—Where the empty cars are to be returned to the cages by the rear of the shaft on a trestle, it is generally done by back-switching, as shown in the tipple tracks in Fig. 947.

Where speed is required, this arrangement is preferable to that of returning the empty cars to the cage on the same side from which the loaded cars are removed, as the loaded car can be pushed off the cage by the empty car as the latter is being pushed on.

2597. In Fig. 947 a track for handling the rock is shown. If there is no rock, the tracks between the shaft and the tipple can be made straight.

Instead of tracks at the rear of the shaft, steel plates can be used for the cars to run on, which will not require as much length as the tracks with their switches.

Where a system of tracks becomes very complicated on the tipple landing, on account of the room and number of switches required for the shifting of the cars, steel plates can be used on the tipple platform in place of the system of tracks. Very little distance is needed in this case between the shaft and the tipple and at the rear of the shaft. The disadvantage of steel plates is in their tendency to wear in grooves, if of inferior material. Further, they are so slippery that it is difficult for the men to get a foothold to push the cars; and the cars do not run upon them as easily as upon rails.

It may be necessary, in some mining operations where small cars are used, to hoist two cars side by side on a wide

cage. In this case a tipple platform laid with plates is preferable, and permits also of putting the empty cars on the cage at the rear of the shaft.

In planning the tracks on the tipple landing, the loaded tracks should be straight and as free as possible from switches, or open points, to avoid the necessity of pushing cars sideways to clear the switches.

Sufficient room should be allowed at the rear of the shaft in case of back-switching, or for the introduction of some mechanical device for the removal of the loaded cars from the cage and the shifting of empty cars thereon.

2598. There are two classes of apparatus for shifting cars, known as **car-shifting devices,** one the invention of Mr. Robert Ramsay, the other that of Mr. F. B. Parrish. In the former a ram pushes the empty car against a loaded car on the cage. The empty car is thus placed on the cage and the loaded car pushed off and continues on a track about 32 feet long, with a fall of 4 inches, to the tipple. After being dumped, the car is switched to an empty track, passing to one side of the shaft. This track is about 50 feet long, having a fall of 8 inches, and lands the empty car on a truck, which is moved up an incline at right angles to the line of the tipple platform, and thus raises the car up the vertical height of 1 foot which it has descended in moving from the cage to the tipple and from there to the truck.

When the empty car is thus brought opposite the loaded car on the cage, the power is applied to the ram automatically, and the process repeated of pushing an empty car against a loaded car on the cage, moving the latter to the dump and placing the former on the cage, in readiness to be lowered.

This apparatus requires a distance of about 28 feet to the rear of the shaft, and a width of about 40 feet for the first 10 feet, and 16 feet in width for the remaining 30.

2599. The apparatus designed by Mr. F. B. Parrish operates as follows:

The platform of the cage inclines towards the tipple, so

that when the car is released from the lock holding it on the cage, it moves towards the tipple on a track about 30 feet long, having a fall of about 1 foot.

A rope is operated, first in one direction and then in the other, by an engine located about 23 feet back of the shaft, and to one side of the platform and below it.

This rope operates in a plane about 6 feet above the tipple platform, and passes from the engine up to an overhead pulley, and then continues by the side of one cage compartment to the side of one of the tipples, and from this tipple to the second one, and then back by the side of the other cage compartment to a pulley 23 feet back of the shaft, and on the other side of the platform from the engine. From this pulley it passes to the engine.

A chain attached to this rope has a hook which is attached to the hinge of the car door, and when the car has been emptied at the tipple the rope is operated to carry the car on the empty track, about 50 feet long, around the side of the shaft, and up an elevation of 18 inches to a truck operated by the same engine, and carrying the empty car opposite the cage. The empty car is then moved on the cage at the same time that the loaded car is removed. The empty car rests on an inclined track on the truck, and at sufficient elevation to move to the cage by gravity when released.

In both these devices, the shifting trucks are arranged so that when one is moving the empty car to the position opposite the cage of one compartment, the other truck is moving to position for receiving the empty car for the other compartment. This apparatus requires a distance of about 24 feet to the rear of the shaft and a width of 32 feet.

2600. If the arrangement of tracks or lack of space prevents the use of the preceding arrangements, mechanical lifts can be introduced at some portion of the return empty tracks, where they are straight, as at the sides of the shaft, in Fig. 947, and the cars can be raised sufficiently high to continue over the back switches at the rear of the shaft.

The usual style of lift is made of sprocket-chain arranged

in the center of the track for a length of 6 or 10 feet, as required, and having a rise of from 1 in 5 to 1 in 3. Lugs are attached to the chain at intervals, which move the car by pushing against its axle, bumpers, or blocks bolted to the bottom of the car. The advantage of car-shifting devices is the increased rapidity with which the cars are handled.

2601. The usual grades with gravitating tracks are as follows:

For loaded cars a steep grade is provided at the start. It is about $\frac{3}{4}$ or $\frac{1}{2}$ inch to the foot for 10 or 15 feet, and the balance of the distance is at the rate of 6 inches per 100 feet for straight track, and more if tracks are uneven and curved.

The return empty track can have a short steep grade of $\frac{3}{4}$ to 1 inch to the foot for the first 10 feet, and at the rate of 9 to 12 inches per 100 feet for the balance of the distance. If the short steep grades produce too great a difference in the elevation of the tracks, they can be omitted, as the cars can be given a start by being pushed by hand.

ARRANGEMENT WHERE THE SHAFT HAS ONE COMMON LANDING LEADING TO THE TIPPLE AND TO THE YARD LEVEL.

2602. It is desirable in this case that the shops, supply houses, and material yards be located conveniently to the return empty track and its branches.

Generally, around or near the air-compartment side of the shaft will be located one or more of the buildings for ventilation, pumping, air-compressors, an electric plant, or an engine for underground haulage. The location for the empty return track, and, therefore, the shops, yard, etc., may have to be, on this account, on the opposite side of the shaft from the air compartment.

If the tracks between the shaft and the tipple are short, their plan will be similar to the arrangements already explained for returning the cars to the cages, either by the front or rear of the shaft, with some modifications for switching cars between the mine and shops and yards by means of turnouts or crossings.

If the distance is long between the shaft and the tipple, the loaded tracks from the cages should be led together at a short distance from the shaft, and one loaded track be used to the tipple.

After dumping at one or more tipples, the empty cars should be switched or crossed over to one empty track returning to the shaft, from which branch tracks turn off to repair-shops, lumber-yard, etc.

Another track with branches should be turned off from the loaded track near the shaft, and run to the yards. Over this track coal is taken to the boilers, and the crippled cars, machinery, and other material from the mine are taken to the shops and the yard.

2603. In Fig. 948 is given an ideal arrangement of tracks in the case of a shaft with only one landing, where the empty cars make a half circuit in returning to the shaft. This plan is preferable to the arrangement of back-switching of cars, provided there is sufficient room outside for the curved tracks and the arrangement of the shaft bottom will permit of its adoption.

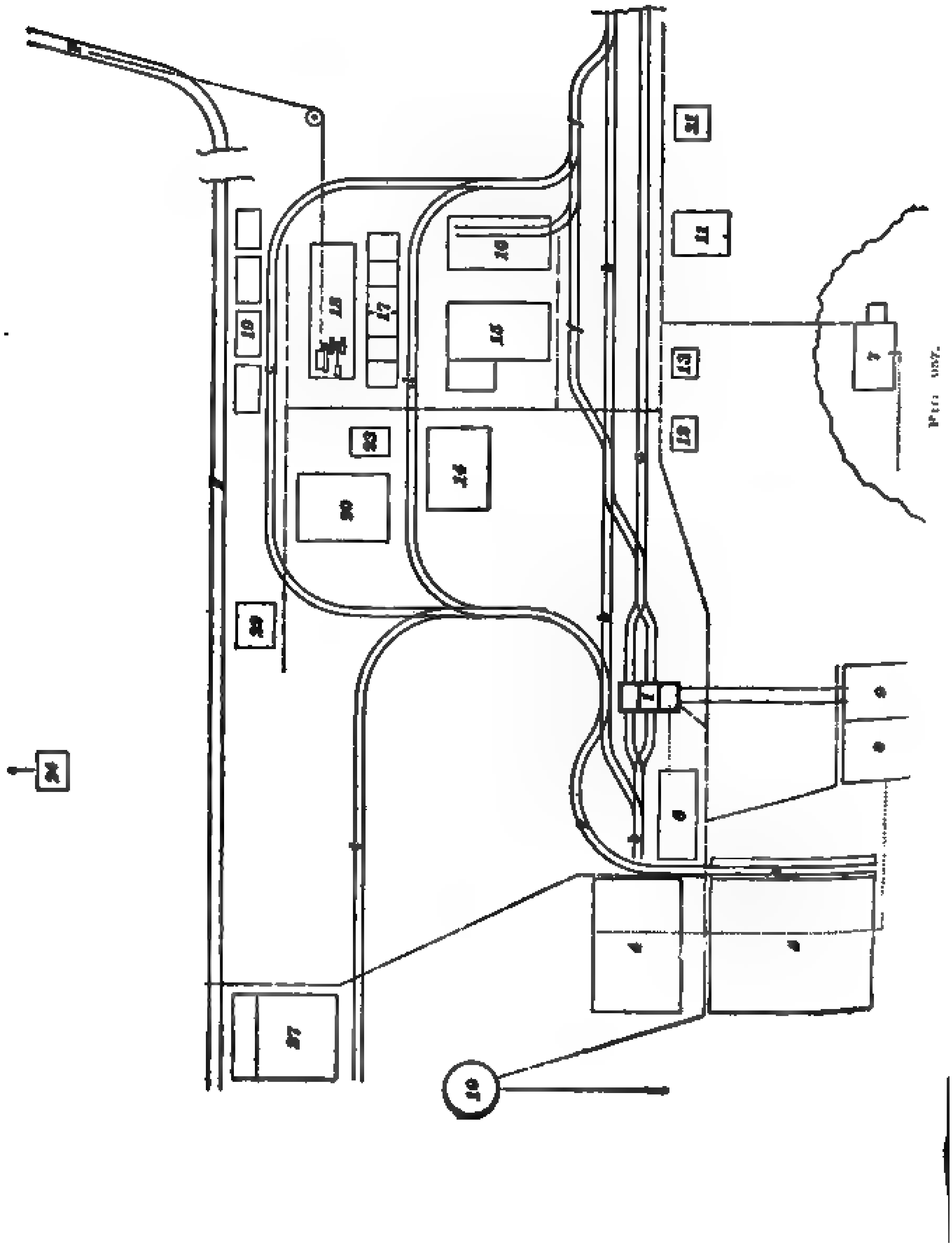
In this plan, the empty cars, on arriving at the rear of the shaft, have their door ends reversed.

This requires the tracks at the shaft bottom to be similar to the arrangement on the outside, so that cars will be headed correctly into the mine.

Some shaft bottoms are planned especially to introduce this arrangement.

If the conditions do not permit of using this system of tracks, they can be made nearly parallel, and arranged for the return of the empties to the mine by the front or rear of the shaft by back-switching, as shown in Fig. 957, and the main features in switches and branch tracks can be still maintained.

2604. In Fig. 957 the shaft is shown at *1*, the scales at *2*, and the tipples at *3*. The hoisting-engines are shown at *4*, the boilers at *5*, the Cornish, or bull, pump at *6*, and the fan, which may be some distance away from the rest of the



plant, is shown at *7*. The electric or compressed-air plant is shown at *8*, and the underground haulage plant is shown at *9*. The wires from the electric plant, or pipes from the compressed-air plants, as well as the ropes from the haulage plant, enter the mine through the pump compartment of the hoisting shaft. The tank for water-supply is shown at *10*, the wash-house at *11*, the mine office at *12*, the lamp house at *13*, the check-clerk's office at *31*, and the weighman's office at *32*. The machine-shop is shown at *14*. The blacksmith shop *15*, with an addition for tools, is shown alongside of the carpenter shop *16*. The sawed-lumber yard *17* is close to the carpenter shop, and adjoining it is the sawmill *18*. The timber and rail yard is shown at *19*. The supply house is shown at *20*, the oil house at *22*, and the iron and pipe shed is shown at *23*, the latter being convenient to both the machine-shop and blacksmith shop. The powder house is shown at *24*. As mentioned in describing other plans, the powder house should be located at least 1,000 feet away from other buildings. The stable *25* and the harness and wagon house *26* are side by side. The hay and feed storehouse *27* is located alongside the material track, with a wharf for receiving feed on the side next the track. It is also close to the stable.

2605. In this arrangement, the loaded cars from the shaft *1* are run over the track *a* to the scales *2*, and thence to the tipples *3*. The empty cars return to the shaft by track *b*. The rock cars are run to the dump over track *o* on one side of the tipples, and are returned to the shaft over track *u* on the other side. The cars of dirty coal are run onto track *l*, and cars needing repairs are run on track *s*. Timber and rails are taken to the shaft over track *c*, and the supplies are taken to the shaft over track *c* or track *d*. Sawed lumber and machinery are moved to any part of the plant by means of track *d*. Empty cars are taken to the yard or to the carpenter shop on track *f*. Hay and feed are taken to the shaft over track *t*. The railroad-tracks are arranged under the tipple, as in Fig. 948. The empty shifting

track is shown at *A*, the lump-coal track at *B*, the nut-coal track at *C*, and the pea-coal track at *D*. The track for material to the timber-yard, oil house, hay and feed store-house, etc., is shown at *F*. If necessary, an inclined plane can be put in this track, up which loaded cars can be pulled by a wire rope attached to a drum run by the saw-mill engine. The location of the rope in such a case is shown as —··——··—. The water-supply pipes are shown thus ——·——·—. The steam-pipes are shown thus ....

2606. In some plants the peculiar position of the tipple with relation to the shaft and other buildings renders the plan of half-circuit tracks more feasible for the return of empty cars to the shaft, and if the reversed position in which they arrive there is not desired, they are turned on plates located at the rear of the shaft before entering the mine.

If the tracks make a half circuit at the tipple and also a half circuit at the rear of the shaft, the cars will have made a full circuit, and arrive at the shaft headed the same way as when they left it.

In any of the arrangements of tracks where the distance is great between the shaft and the tipple, they can be made gravitating, and mechanical lifts introduced at some one or more points where the empty and other cars all pass over the same track in returning to the mine, as shown in Fig. 948.

If the difference in the elevations of the various tracks is very great and occurs in the yard, it may prove a disadvantage, so that it is preferable that the mechanical lift in this case be near the tipple, and the elevation of the yard be kept nearly level.

A second mechanical lift may be necessary at the rear of the shaft to overcome some slight differences of elevation in the track, if the turnout from the loaded track to the yard is made gravitating.

The location of a mechanical lift at the rear of a shaft is objectionable unless other tracks exist, so that heavy machinery, lumber, etc., will not have to be handled over it.

If it is desired to maintain the system of tracks between

the shaft and tipple at a level, an endless rope can be introduced, moving in the direction of the loads and to one side or over the cars, and returning in the direction of the empty track.

YARD TRACKS.

2607. Branch tracks, conveniently arranged according to the conditions and requirements, will be needed in the yard, in the several directions, as shown in Figs. 947, 948, and 957, for the following purposes:

1. Tracks for handling heavy timbers, rails, piping, etc., which will have to be transferred to the cages. (See tracks *c*, Figs. 947, 948, and 957.)

2. Either the same or other tracks for loading sawed lumber, props, hay, feed, etc., onto cars going into the mine from various storehouses. These cars will generally run directly onto the cages. (See tracks *d*, Figs. 947, 948, and 957.)

3. Tracks to the carpenter shop for cars needing repairs. (See tracks *d*, Figs. 947 and 948, and track *f*, Fig. 957.)

4. Track to the machine-shop for handling machinery on trucks from the various points above and below ground. (Track *g*, Fig. 947, track *e*, Fig. 948, and track *d*, Fig. 957.)

5. Tracks to the boilers for coal, unless the boilers are handy to railroad-tracks and can be more conveniently supplied from railroad-cars loaded with coal or screenings. (Tracks *e*, Figs. 947, 948, and 957.)

If there are isolated plants using boilers, coal should be supplied thereto by branches from the mine or railroad-tracks as most convenient.

SPECIAL FEATURES IN BUILDINGS AT SHAFT MINES.

2608. The arrangements and features of each of the buildings in the mine plant will be explained farther on, but a few important points in their arrangement at shaft mines should be mentioned here.

The danger in case of fire, where timber is used in the construction of many of the buildings that of necessity are so near the shaft, makes it advisable that the head-frame,

platform, and structure be built of iron or steel, also the tipple structure if it is near by. All roofs should be of corrugated iron.

If the engine and boiler houses have corrugated roofs, the greatest danger in case of fire there is overcome, but greater safety is secured if their walls also are built of stone, brick, or corrugated iron.

If a fan house is located near the third compartment of the shaft, it should also be made fireproof with brick walls and iron casing and roof.

The carpenter, blacksmith, and machine shop and other buildings can be sufficiently removed so as not to be sources of danger in case of fire; although, if the cost of construction can be incurred, they should be made fireproof. In the latter cases, the risk of the loss of each building and consequent delays to the operations are principally concerned.

With wooden buildings surrounding the shaft opening, there is risk of the total destruction of the property, inside and outside, as well as the possible loss of life in case of fire.

VENTILATING PLANT.

2609. As there are generally two openings to a shaft-mine, it is preferable that the ventilating plant be erected at the second opening and not at the third compartment of the hoisting shaft, so as to leave this compartment open as an air inlet, also for pipes or rods for pumping, ropes for haulage, or air-pipes, or electric wires for haulage, coal cutting and pumping underground, as the case may require.

It is well to have a fan erected near the third compartment for use in case of emergency, and connected therewith by an underground conduit just below the surface. (See Fig. 947.) The air compartment can remain open, free from obstructions, until it should be necessary to use the fan, when, by closing certain doors, always in readiness, at the top and bottom of the shaft, the ventilation can be controlled as may be desired, in case of accident to the other fan or in other emergencies.

SURFACE ARRANGEMENTS AT A MINE OPENED AT A POINT BELOW THE TIPPLE LEVEL.

GENERAL PLAN.

2610. The distinctive features wherein a mine opened at a point below the level of the tipple differs from mines opened otherwise, are in the arrangements necessary for hoisting trips of cars with ropes on an incline more or less steep and long, and finally landing on a nearly level platform, where they are dumped; also in the connection of the slope and yard tracks, more especially where the tipple is at a higher elevation than the yard tracks, and cars must be lowered or hoisted by the slope engine in shifting them between the slope and yard tracks.

Sometimes but one or two cars are hoisted at a time, due to a short haul, requiring very little power and small landing room. Generally from six to fifteen cars are hoisted at a time, requiring long landing room and powerful engines, which make it important that the latter be located on line with the slope and trestle at some convenient point, so that the structure can be properly braced, and no complicated strain brought to bear on the sheaves at the knuckle and points where the rope changes direction.

In this particular the location of the engine on line with the slope is, perhaps, of more importance than at a drift mine, because at a drift mine either the power required for the haulage of long trips is not so great as at the slope or the surface location of the tracks, where curves are required, permit of a solid foundation for the sheaves, so that the ropes can be led to the engine, whether located on line with the drift or not.

2611. A slope mine differs from a shaft mine, but is similar to a drift mine in the following respects:

1. The lowering and hoisting of men and mules is not necessary unless the slope is very steep, as a second opening

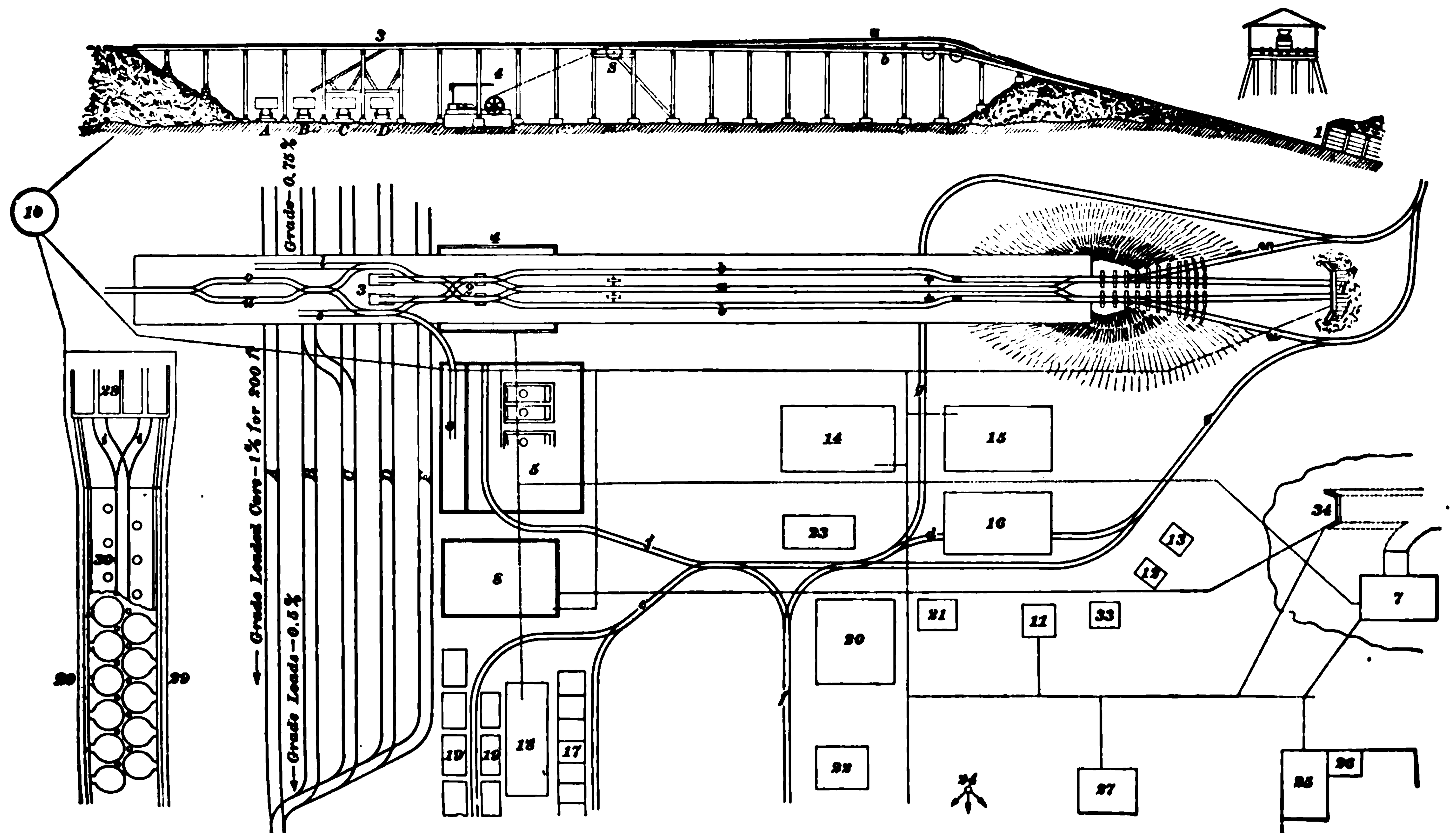


FIG. 958.

is generally provided for their entrance and outlet, although in some cases they enter by the haulage road.

2. Stables for mules are not usually built underground, in which case it is not necessary that hay or feed be lowered into the mine, unless it is for the noon-day meal.

3. Long timbers, pipe, rails, etc., can be conveniently lowered into the mine, whereas at a shaft mine, unless proper provision is made at the top and bottom and cages are conveniently arranged, it is sometimes inconvenient, if not impossible, to handle material of lengths as great as desired.

2612. A general arrangement for a slope is shown in Fig. 958. In this case the landing platform and tipple is between the mine outlet and the railroad-tracks, and the ground is comparatively level, with the mine outlet near the trestle. In this figure, which shows a plan of the outside arrangements, with an elevation of the trestle and a cross-section of the trestle at the knuckle, the mouth of the slope is shown at *1*, with an extension of its grade to a point high enough for a trestle with height enough above the railroad-tracks for the tipple. The scales are shown on the trestle platform at *2*, directly over the engines *4*, and the dumps are shown at *3*. The boiler house, with coal-bins in front, is shown at *5*. The compressed-air or electric plant is shown at *8*. The tank for water-supply is shown at *10*, and the water-supply pipes are shown thus — - — - —. The wash-house is shown at *11*, the mine office at *12*, and the lamp house at *13*. The machine-shop *14*, blacksmith shop *15*, carpenter shop *16*, and iron and pipe house *23* are shown in a group, which is an arrangement that is most convenient. The sawmill *18*, timber-yard *19*, and sawed-lumber yard *17* are also conveniently grouped. The supply house *20*, oil house *22*, and the clerk's office *21* are also conveniently located. The powder house *24* is not shown, but arrows indicate its approximate location, about 1,000 feet from all other buildings. The stables *25*, harness and wagon house *26*, and hay and feed storehouse *27* are located as near each other as possible. The manway or traveling way *34* is shown in connection

with the fan *7*. These may be located several hundred yards away, in which case a separate steam plant will be required for the fan. If the manway and fan are located but a short distance off, steam can be supplied to the fan-engine from the main boiler plant *5*. Or, if far away and there is an electric plant, the fan may be run by a motor, fed by wires from the dynamos. The steam-pipes from the boilers are shown thus......... The coal-bins for coke-oven supply are shown at *28*, the coke-ovens at *30*, and the coke wharves at *29*.

There are a few exceptions to this general arrangement, which will be referred to farther on.

The length of the trestle platform, as shown in Fig. 958, will depend upon the number of cars to be hoisted and the amount of shifting of cars necessary, also whether rock is to be handled.

2613. The system of tracks is arranged so that the loaded cars from the mine are drawn up and over the knuckle and on to a center track *a*, which is slightly elevated, so as to have a down grade of 1.66% towards the tipple. The loads are then distributed to the tipples, and when dumped they are run to either of the empty tracks *b* on the outside of the loaded tracks, which have a down grade of 2% towards the knuckle to facilitate their movement. The cars are collected from the knuckle towards the engine and held by a "stop" at the knuckle.

A rope is attached to the rear of the trip, and when it is to be lowered the "stop" is removed, and the incline of the tracks is sufficient to start the cars towards the slope.

In the passing of the loads from the slope tracks to the high center elevated track, there is some friction of the ropes against the side of the elevated structure, requiring guides or sliding pieces made of wood and laid from the inside rail of the empty tracks to the outside of the rails of the loaded tracks, so as to reduce the wear on the rope and guide its movement up from the sheaves and to one side of its slope line. The empty cars, in returning to the mine, carry the rope to the proper center line before passing over the knuckle.

In this arrangement the lead between the sheaves and the point where the loads turn out to the center track should be long. If the angle at the knuckle is too great, drums are better than sheaves at the knuckle, for the ropes will leave them more readily, as the loaded trips pull the rope sideways.

In this system of three tracks, sufficient length is provided for accumulating two trips on each side. The rope lies between the tracks from the knuckle to the sheaves, where the rope passes down to the engine, so as to be out of the way in shifting cars, and not in the center of the tracks.

Sufficient distance should be allowed from the engine to the sheaves *S*, deflecting the ropes from the platform to it, so that the coiling on the drum will be perfect. This distance depends upon the width of the drum and the load; 45 feet will be sufficient with a small drum and light load, although the longer the distance the better. One hundred feet is sufficient in all cases, and if this distance is needed and can not be obtained in the direction of the engine from the sheave *S*, shown in Fig. 958, the engine can be located under the trestle in the opposite direction from the sheave.

2614. An examination of the plan of tracks on Fig. 958 will show that loaded rock cars are run to the dump over track *o*, and the empty cars are returned over track *u*. Dirty coal is switched on to track *l* and crippled cars are switched on to track *s*. Coal for the boilers is run over track *e*. Cars are hoisted from and lowered to the yard level on tracks *w* by the hoisting-engine. Timber, lumber, and rails are taken to the slope over track *c*. Cars are taken to and from the carpenter shop over track *d*. Ashes are taken from the boiler house to the ash dump over track *f*. All of these tracks connect with tracks *g*, which in turn connect with tracks *w*. The larry tracks from the coal-bins *28* to the coke-ovens *30* are marked *i*.

The railroad-tracks are arranged as follows: *A*, empty shifting track; *B*, lump-coal track; *C*, nut-coal track; *D*, slack track, and *F*, material track, or track on which supplies in carload lots are received.

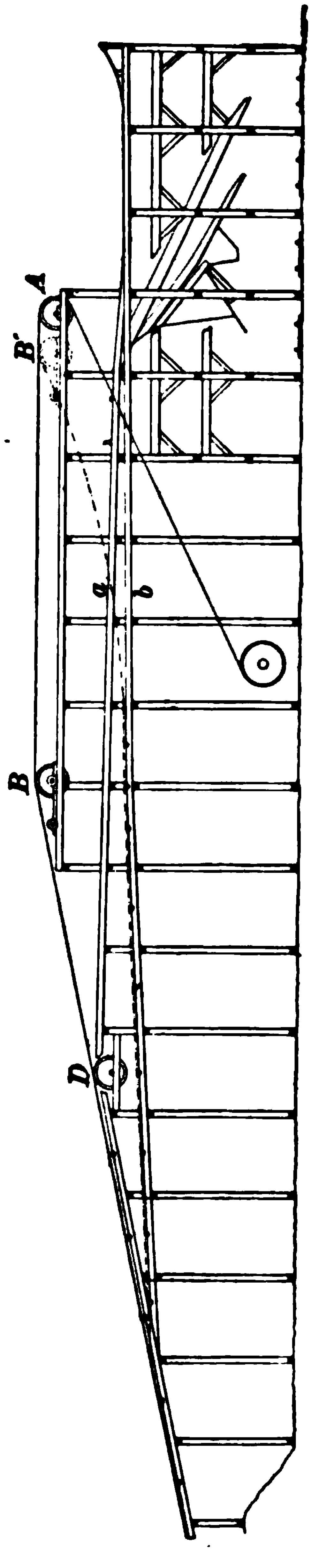


FIG. 959.

SPECIAL ARRANGEMENTS.

TRACKS.

2615. Fig. 959 shows an arrangement of tracks wherein an overhead traveling pulley is employed to guide the rope in raising the loaded cars to an elevated track which has a down grade to the tipple, and for returning the empty cars accumulating on an empty track with a downgrade to the slope. The empty track switches into the straight slope track, similar to the arrangement in Fig. 958.

In this arrangement the sheaves *A* and *D* are stationary. The sheave *B* is mounted on a truck which moves along a level track. When starting to hoist the load, the truck and its sheave are in the position *B*, but by the time the load has passed the knuckle and is running along the loaded track *a* the truck has moved to the position *B'*. When the empty cars on track *b* are attached to the rope and they are lowered, the truck leaves the position *B'* and runs along to the position *B*. The loaded track *a* has a fall of 1.66% towards the dump, and the empty track *b* has a down grade from the dump.

PUMPING AT A SLOPE MINE.

2616. The advancing face of the slope or dip workings requires that the pumping plant be continually extended with the increasing depth

of the mine. A very common practice has been to carry a line of steam-pipe from the boilers on the surface down the slope, or by a separate pipe-way, to a steam-pump located underground. This sometimes results in steam doing much damage, and it is a source of danger.

If a power is to be carried from the plant near the tipple to distant points underground for pumping, either compressed air or electricity should be used, and not steam.

In order to avoid the great length of pipes in long slopes and reduce the expense of pumping, it is best to bore two holes or sink a shaft from the surface to meet the face of the dip workings, and erect a pumping plant on the surface at that point.

In this case power can also be transmitted through the bore-holes or shaft for use in pumping, hauling, or coal cutting at distant points underground. Provision should be made in this case for supplying the plant with coal, carried thereto on mine or railroad cars, or by a rope tramway.

ARRANGEMENTS OF SURFACE WORKS FOR SPECIAL CONDITIONS.

2617. In some instances the distance between the trestle and the mouth of the drift or slope mine may be very long. In this case it will be found more convenient to erect most of the shops, storerooms, stables, material yard and offices near the mine opening, as in the surface arrangements at a drift mine. The location of the machine-shop will depend upon whether most of the machine-work is inside the mine or on the surface.

If the slope enters the hillside under such conditions that there is considerable ground at the same elevation as the platform of the tipple, and readily reached by level tracks, the shops may be erected thereon, and tracks run to them from the empty tracks on the tipple platform. The location of the storerooms and material yard should be at such points as are most convenient. If at a lower level, they can be reached by an inclined track or wagon road. This

arrangement is shown in Fig. 960, with a system of tipple tracks that is suitable for any slope. The tracks, however, are not gravitating.

2618. In Fig. 960 the mouth of the slope is shown at *1*, the engine under the platform at *4*, and the sheaves for

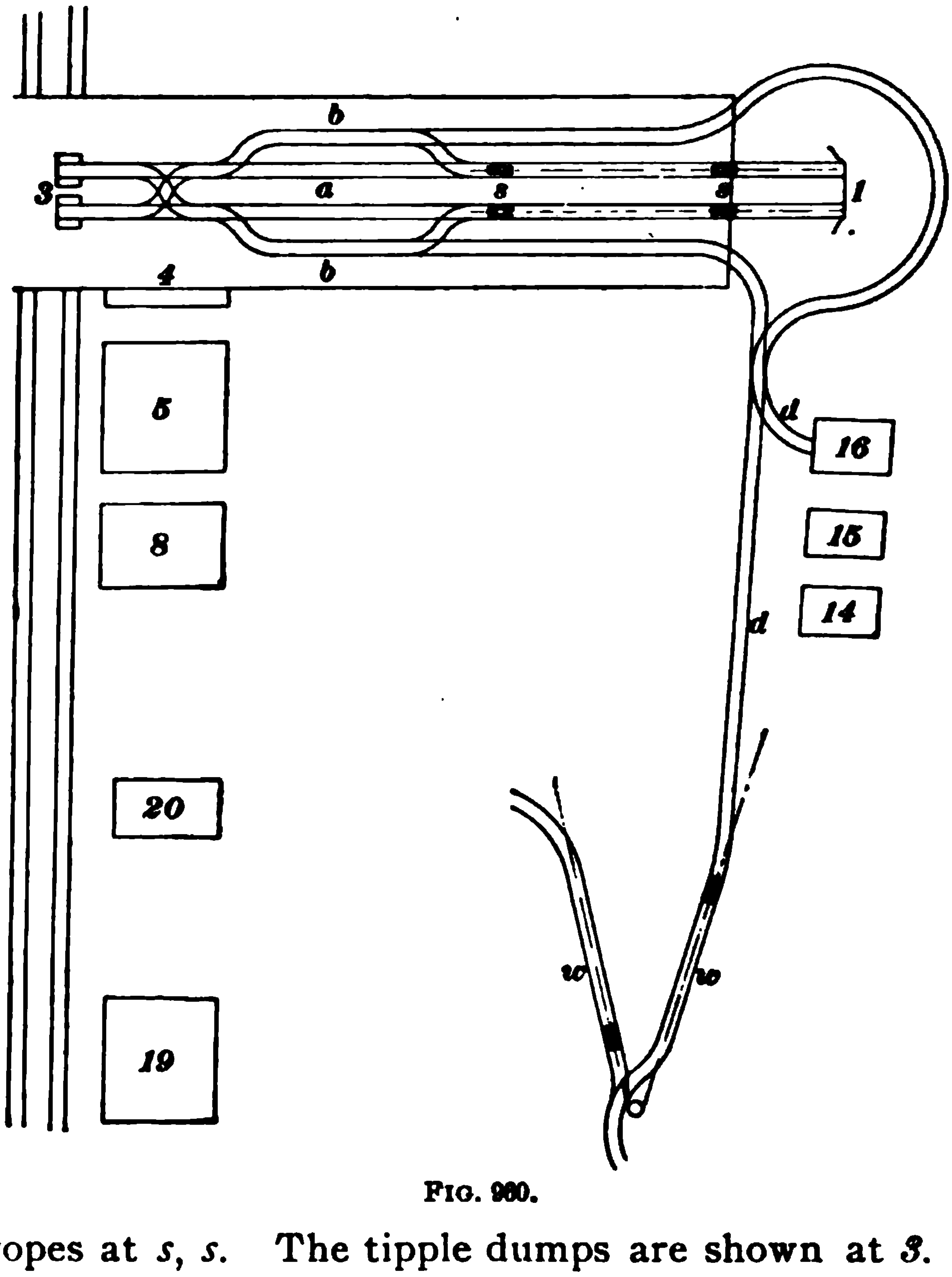


FIG. 960.

the ropes at *s*, *s*. The tipple dumps are shown at *3*. The boilers are shown at *5*, the compressed-air or electric plant at *8*, the supply house at *20*, and the lumber and material yard at *19*; these are all located convenient to the railroad. The carpenter shop *16*, the blacksmith shop *15*, and the machine-shop *14* are convenient to the slope and at a suitable height to have a level track *d* run from them to the platform. The loaded cars from the slope are run over tracks

a to the tipple, and the empty cars are returned to the slope over tracks *b*. The inclined tracks *w* are used to connect that portion of the plant near the railroad with the trestle and the buildings on the hillside. The cars are hoisted up the inclines by a rope from a drum in the machine-shop. This rope is shown thus — - - — - - —.

The inclined tracks *w* to the yard and buildings at the lower level, for hoisting lumber, etc., can be arranged near the slope and be operated by the slope rope by introducing sheaves, etc., for its deflection to the line of the inclined tracks.

The relative location of the railroad-track with respect to the line of the slope may be such that it will require considerable curvature in the tracks to bring the coal to the dumping point. The center line of the engine and that part of the platform supporting the ropes can be kept on line with the center line of the slope, and from the sheaves or the knuckle the alinement of the tracks and trestle can be given the necessary curvature to reach the railroad-tracks. Or, if the railroad-tracks lie beyond the slope from the engine, tracks can be led to a tipple by back-switching the cars after they land on the platform.

2619. The trestle should not be made to cross the mouth of the slope unless built of fire-proof material, so as to avoid danger in case of fire. If the distance is short between the railroad-tracks and the knuckle, the engine can be located and the tipple and chute arranged as shown in Fig. 961 in plan and elevation.

The points in common on both plan and elevation are the mouth of the mine *1*, the knuckle *2*, the sheaves for deflecting the rope *3*, and the tipples *4*. The location of the engine is shown at *5*, and the slack, nut, and lump coal tracks are shown at *C*, *B*, and *A*, respectively.

The loaded tracks from the slope to the tipples are shown at *b*, *b*, and the empty tracks at *a*, *a*. The inclined tracks to the yard are shown at *c*, *c*.

In some slope arrangements the grade is so steep that it is

necessary to employ a device called a "dummy," or platform car, for raising the car. Only one car is hoisted at a time.

This arrangement will be explained under inclined planes,

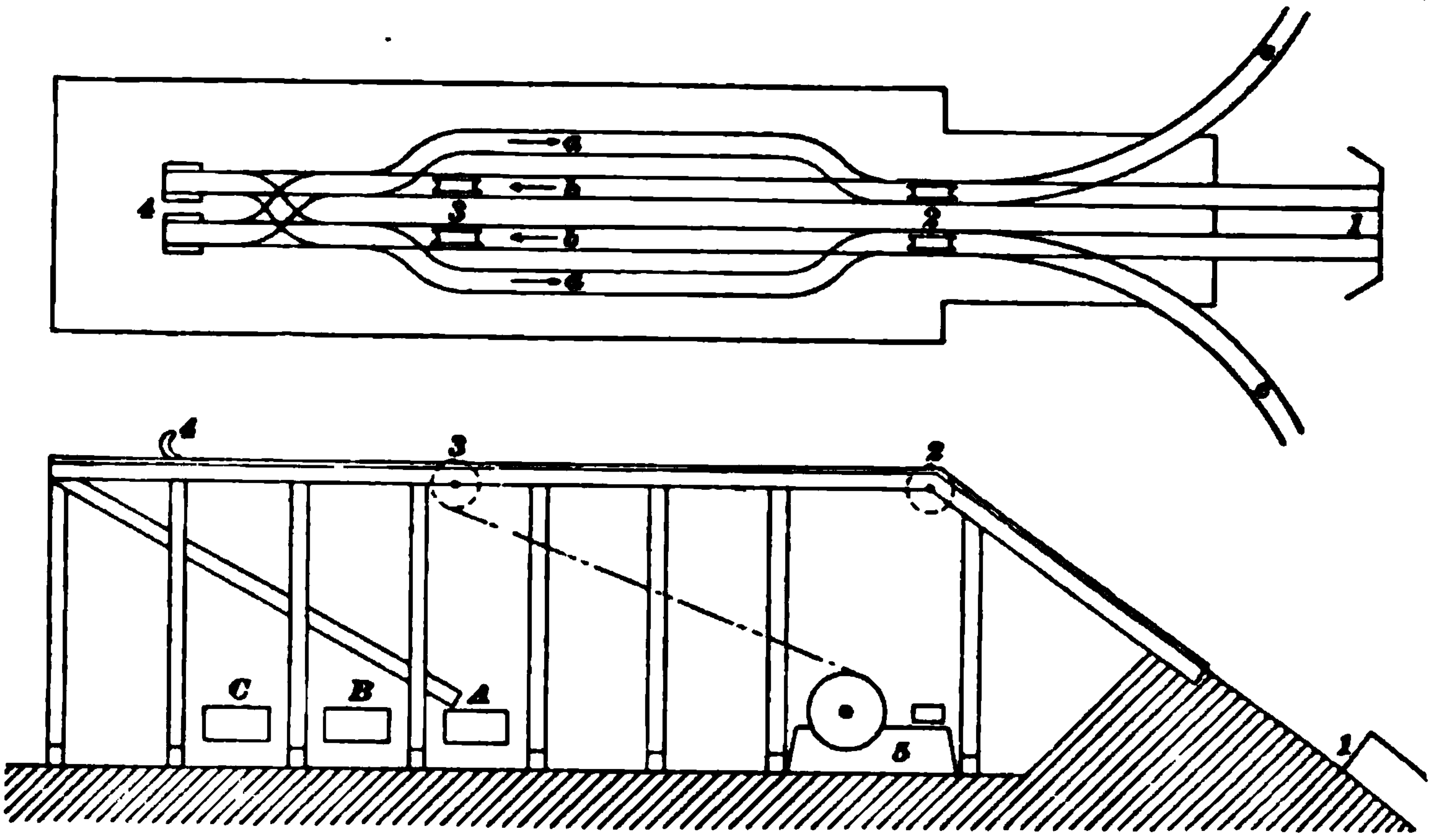


FIG. 961.

as its operation is very similar, whether coal be lowered or hoisted.

2620. The minimum grade on which empty cars will return of their own weight, and draw the rope along with them, depends upon the number of cars in the trip, and resistances at the engine, and friction of the rope on the roadway and rollers.

Under certain conditions, empty cars will descend on a grade of 3%, but the resistance met with in slopes makes it unwise to depend upon the ordinary types of slope haulage engines for grades less than 5%. Even with 5% grade, the slope should not be more than 1,000 feet long. There should be four or more cars in a trip, and loaded cars should not be hoisted at the same time. The trip should be started with some headway at the knuckle.

It is better to have the grade 7% to 8%, and for slopes longer than 1,000 feet this should be the grade for the balance of the distance, when the grade is 5% for the first 1,000 feet. For less grades than the above, the engine should be arranged for operating as an endless or tail rope engine.

SURFACE ARRANGEMENTS AT MINES OPENED BY DRIFTS.

ARRANGEMENTS ON A HILLSIDE, WHERE CARS CAN BE RUN BY GRAVITY TO THE TIPPLE.

2621. The particular features wherein a mine opened as above differs from other mine openings are:

1. In the movement of the coal to the outside.

This is done by a haulage way to the outlet over level or nearly level roads, for which either mules, endless or tail rope haulage, steam, electric, or compressed-air locomotives are used.

The haulage may be in trips of from one to ten cars with mules, and from about twenty to forty cars with rope or locomotives, although in some instances more cars are hauled.

2. In the drainage of the mine, which seldom requires the handling of water to any great height, as it will usually flow to the outside by the gradual fall in that direction. If it does not, the mine can be drained by the arrangement of siphons, or of light-pressure pumps, for raising water to small heights from local swamps or dips.

2622. The usual height of mine openings, opened as above, and near the dumping point, should be from 20 to 35 feet above the railroad-track.

The height of a railroad-car being about 7 to 9 feet above the rail requires that the tipple platform be 11 to 13 feet above the railroad, in which case no gravity system of screening can be arranged. In this case the coal is usually dumped directly into the railroad-car.

Short screens can be arranged with tipples 16 feet high, but for perfect screening, either revolving or shaking screens will have to be introduced for this height. Where the drift is lower than the tipple platform, it should be arranged, if possible, to locate the tipple and drift opening some distance apart, so as to haul the loaded cars up grade by mules, rope, or motors to the height desired for the

tipple. If this is not possible, and where the drift opening is 10 feet or more below the tipple platform, an incline should be introduced to raise the cars to the tipple.

The height of the tipple may be greater than 35 feet, but it is preferable to avoid too long a chute and to introduce special appliances described farther on; or, if there is sufficient height, the mine may be operated with a gravity plane.

Where the coal is dumped into bins for coke-ovens, the tipple may be 60 feet above the railroad-tracks.

The short time and small outlay necessary in opening a drift mine frequently results in the location and arrangement of the surface works for the immediate needs of the operation, and without a view as to the possible requirements of the future.

2623. If the arrangements and their location are of a temporary nature, no serious drawbacks will be experienced, as they can be readily removed or altered, if improvement is found necessary, with increasing and more extended operations.

If, however, one structure after another is added and enlargements made to satisfy temporary needs, the arrangement is apt to assume a permanent character, which in the usual locations of drift openings, due to the limited ground between the hillside and the railroad-tracks, makes alterations and rearrangement of the plant difficult. More especially is this the case if, in addition to the mine plant, coke-ovens or coal-washing plants have been erected in the neighborhood, and are also to be enlarged.

Therefore, due consideration should be given to the planning of the arrangement at the start, so as to early provide for future possibilities. It is important in this respect that the mine opening be straight and properly located, so as to facilitate haulage.

Plenty of room should be allowed between the mine outlet and the tipple for tracks and sidings.

In the early operations, the tipple may be built on line with the drift when the ground is limited between them,

but is sufficient for the length of tracks needed at the opening of the mine.

If longer tracks are needed in the future, it may be necessary to erect the tipple at some distance to the left or right of the drift, and carry the tracks thereto, with curves at the outlet and also at the tipple. It is well to keep in view this possible change in the arrangement, so that permanent structures will not be erected on the ground that may be needed for this purpose, and so that shops and buildings erected conveniently with respect to the first arrangement will also be handy to any future improvement.

ARRANGEMENT OF TRACKS.

2624. A general arrangement of the surface works at a drift mine is given in Fig. 962. The tracks from the outlet to the tipple and at the tipple are shown developed for a system of rope haulage, and for a case where there is considerable landing room between the outlet and the tipple, which permits of locating the haulage engine on line with the drift and only slightly curving the tracks.

The tracks may be made straight and the engine room built beneath the trestle, or if the ground is limited between the drift and the tipple, the engine could be located to one side of the drift and ropes led thereto, by curving horizontally at sheaves erected at a proper point.

The same system of tracks could be used for mule haulage, although a complete arrangement of two tracks, one for loaded cars and the other for empty cars, would be sufficient in this case.

2625. The principal feature in the arrangement of the buildings at a drift mine is the requirement that they be generally located along the contour of the hillside, with very narrow and long yard room. In Fig. 962 the arrangement shows the buildings, yards, and outside tracks on a level with the haulage tracks from the mine.

The boilers and engines should be built either on a level with the mine or railroad-tracks, as the extent and slope of

the ground as well as other conditions may require. But in any event they should be located near the one or the other.

If the conditions are such as to permit of all the buildings being erected at the yard level of the mine, it will prove the most satisfactory for concentration of the work. This can generally be arranged if small amounts of supplies and material are received by railroad and their handling is not a difficult matter, and if lumber, props, etc., are cut on the surface of the property, or near by, and are delivered close to the mouth of the mine by teams or a tram-road. Or, if the material and supplies for the mine are received mostly by railroad, an inclined road can be built, as shown in Fig. 962 at F', for hoisting railroad-cars to the mine yard level.

2626. As in other general plans, there are more improvements shown in Fig. 962 than are generally necessary at any one mine. The object of this is to show how any of them that may be necessary can be arranged. The mouth of the drift is shown at *1*, the scales at the tipple and also at the coal-bins for coking coal are shown at *2*, the main tipples for shipping coal at *3*, the haulage engines at *9*, the fan at *7*, the compressed-air or electric plant at *8*, and the boiler house with coal-bin in front at *5*. The tank for the water-supply is shown at *10*, and the supply-pipe lines from it are shown thus ——·——·——. The mine office is shown at *12*, and the lamp house at *13*. The machine-shop *14*, blacksmith shop *15*, and carpenter shop *16* are shown in close proximity to each other and to the sawmill *18*, sawed-lumber yard *17*, timber-yard *19*, and the iron and pipe shed *23*. The locomotive, electric or air, that may be used in distant workings beyond the rope haulage is housed in a building *36* adjoining the machine-shop. It enters the mine through opening *35*, over track *c*. The supply house *20* and supply-clerk's office *21* are shown together. The oil house *22* is shown at a point on the line of the supply track F', convenient for the reception of oil from railroad-cars. The stable *25*, wagon and harness house *26*, and the hay and feed storehouse *27* are located close together. The airway

for the mine is shown at *34*. The coal-bins for the two banks of coke-ovens are shown at *28*. The ovens are marked *30* and the coke wharves *29*. At *33* is located a hoisting-engine to hoist cars up an incline from the drift to the level of the platform on the coal-bins, if the topography makes such an arrangement necessary. The weighman's office is marked *31*. The ropes from the haulage engines *9* to the drift mouth are shown thus ——··——··——, as are also those from the sawmill to the inclined tracks *F'*, over which railroad-cars loaded with supplies can be raised to the level of the storehouses, yards, and shops. The steam-pipes are marked ....

The mine-car tracks are marked as follows: *a*, loaded tracks from drift mouth to tipples or coking-coal bins; *b*, empty tracks from tipples and coal-bins to drift mouth; *c*, track to blacksmith-shop, machine-shop, sawed-lumber yard, etc.; *d*, track to carpenter shop and timber and rail yard; *e*, track for boiler coal; *o*, track for rock cars to dump; *l*, track on which cars of dirty coal are switched; *h*, track for larries to coke-ovens. The railroad-tracks are marked as follows: *A*, empty shifting tracks; *B*, lump-coal track; *C*, nut-coal track; *D*, slack track; *F*, material or supply track, which is inclined at *F'*; *G*, coke-car loading tracks, and *H*, empty coke-car shifting track.

2627. Conditions may prevail which make it most advantageous to locate the material yard, storeroom, etc., near the railroad-tracks to avoid delay in unloading material, and to locate the carpenter, blacksmith, and possibly the machine-shop at the mine yard level.

Any hauling of material from the railroad to the mine can then be done over a wagon road or inclined tracks for railroad or mine cars, as may be found convenient.

The stable and feed room should be located at such a point as will be convenient for the traveling of mules to and from the mine, without their having to pass over any haulage roads that are fully occupied with cars and ropes. This is especially the case if the ground is steep back of the drift

mouth, and a roadway can not be constructed there from one side of the drift to the other.

2628. Figs. 963 and 964 show arrangements of tracks and engine where the space between the drift mouth and the tipple is limited. Fig. 963 shows the engine located to one side of the drift, and a device for storing or holding empty

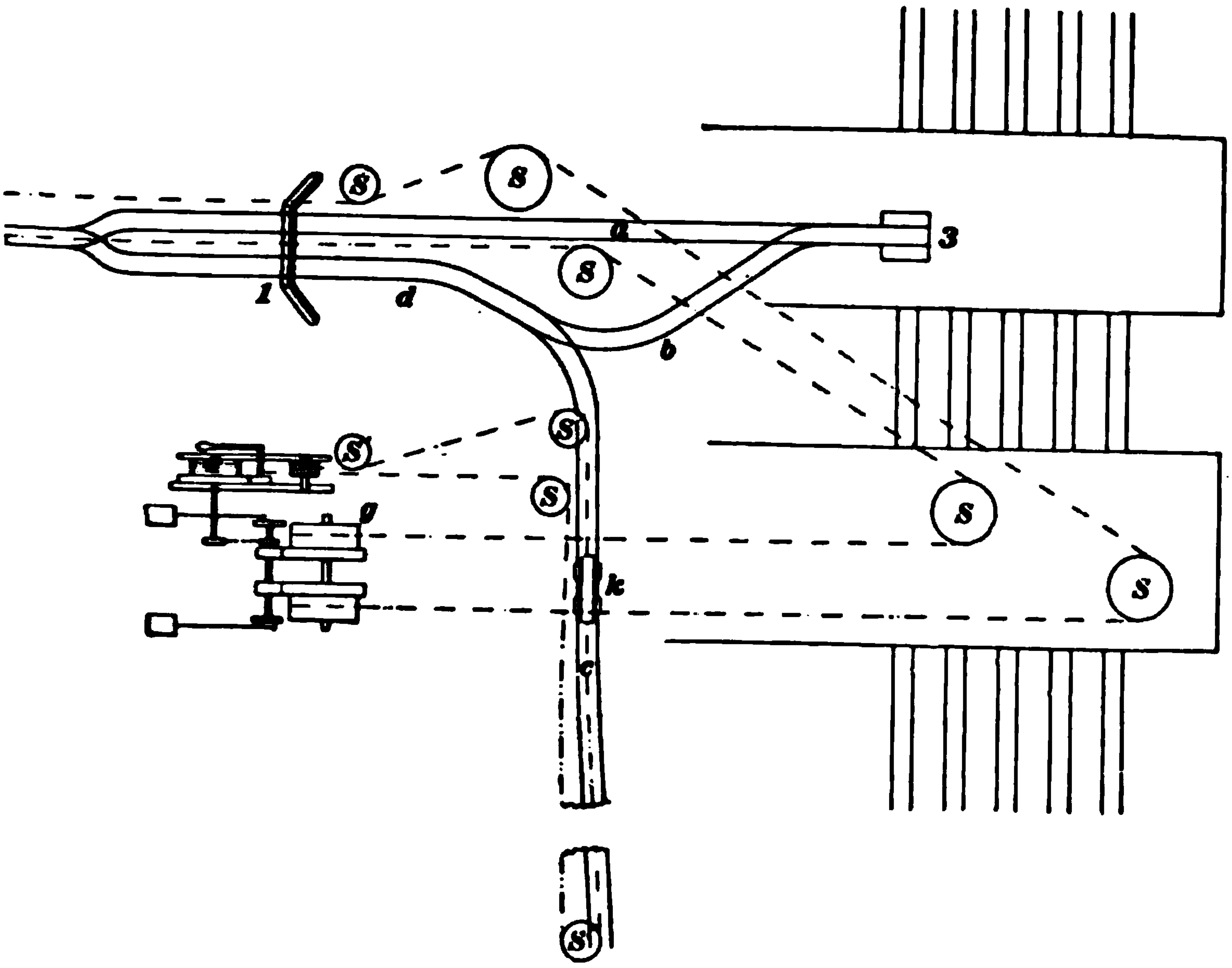


FIG. 963.

cars outside at a plant where the locations of the drift outlet and tipple have become fixed.

In this case the loaded track *a* has a fall of 1 ft. in 100 ft. from the drift mouth to the tipple *3*, and the empty track *b* has a similar fall from its junction with the loaded track *a* to its junction with the storage track *c*. This storage track *c* has a fall of 16 in. per 100 ft. away from the drift mouth from the point *d*. By referring to the figure, it will be seen that the tail-rope haulage is operated by the ropes running around the large sheaves *S*, and that there is an endless-rope

mechanism attached to the engines by a friction-clutch which actuates the endless rope around sheaves S', by means of which and the dummy k the empty trip is made up.

2629. Fig. 964 shows a tipple located at some distance and to one side of the line of the drift, so as to provide necessary length of tracks outside.

In this case the drift mouth is shown at *1*, the tipples at *3*, the tail-rope engines at *9*, and the boilers at *5*. The sheaves around which the main ropes run are marked s, and

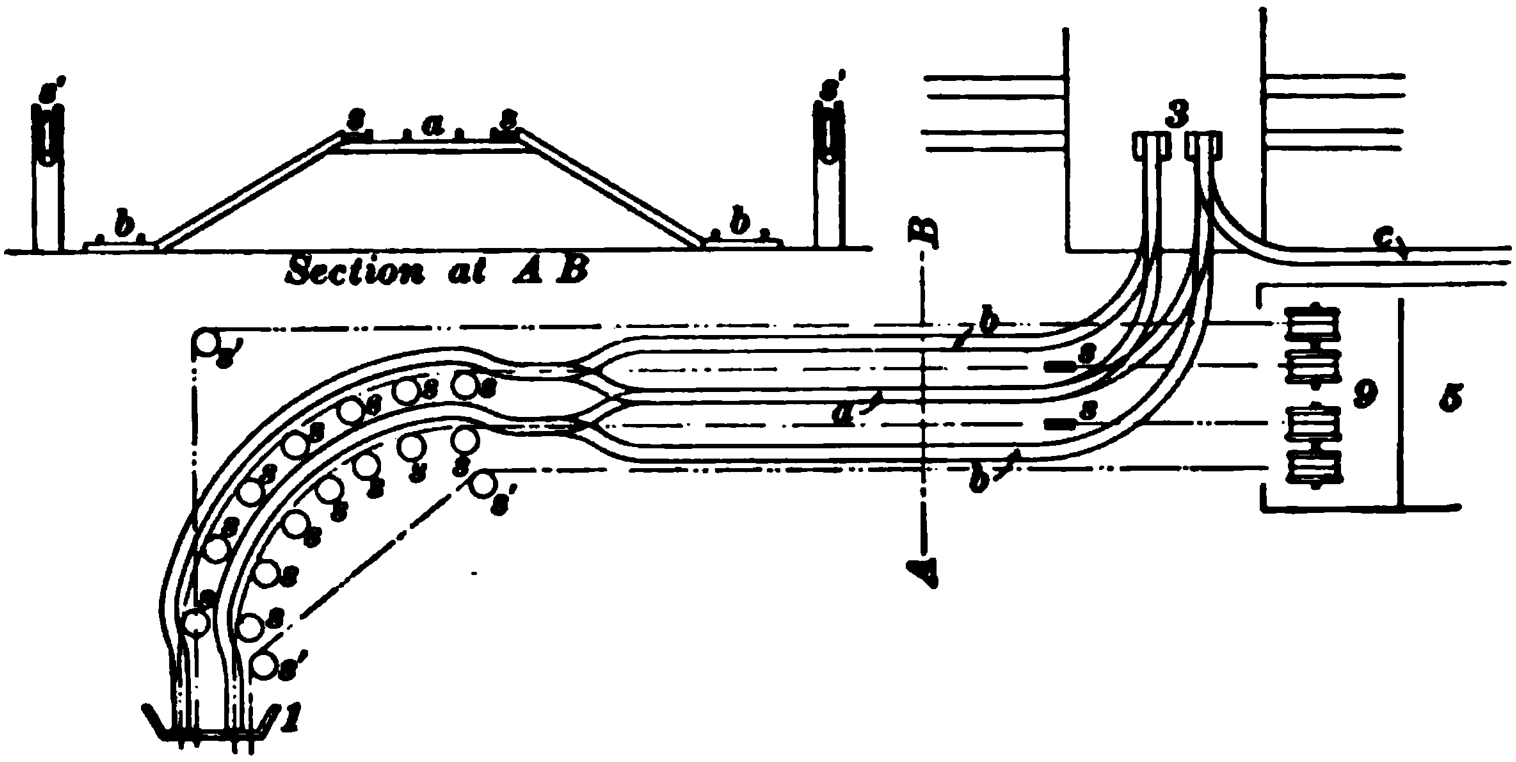


FIG. 964.

those around which the tail-ropes run are marked s'. The loaded tracks from the drift mouth to the tipples are marked a, the empty tracks b, and the track to the shops and yard c.

The surface works should be located handy to the return empty tracks, so that cars can be switched out and back without crossing the loaded track.

In the case of the two outside tracks being the empty ones, as shown in Fig. 962, cars from one side can be shifted to the desired side by switching around the tipple.

Tracks for coal to the boiler should also be provided. In Fig. 962 the coal to the boilers and the rock is switched out on the track between the two tipples, and the coal is back-switched to the boilers over track e.

SURFACE ARRANGEMENTS AT A MINE OPENED BY A DRIFT ON THE MOUNTAIN-SIDE ABOVE THE TIPPLE LEVEL.

2630. The main features connected with a mine opened above the level of the tipple are in the arrangements for lowering the coal from the mine outlet to the dumping platform.

Sometimes the coal is dumped from the cars at a tipple on a level with the mine outlet into a long chute reaching to the loading point below. The arrangement of the surface works around the mine outlet in this case would be similar to that of those connected with a drift mine.

The use of a long chute is, however, objectionable, as the coal grinds and is broken by the force with which it strikes the chute and gates.

A height of about 25 to 30 feet above the tipple tracks should be the minimum for the adoption of a gravity plane when its length is 300 feet or more.

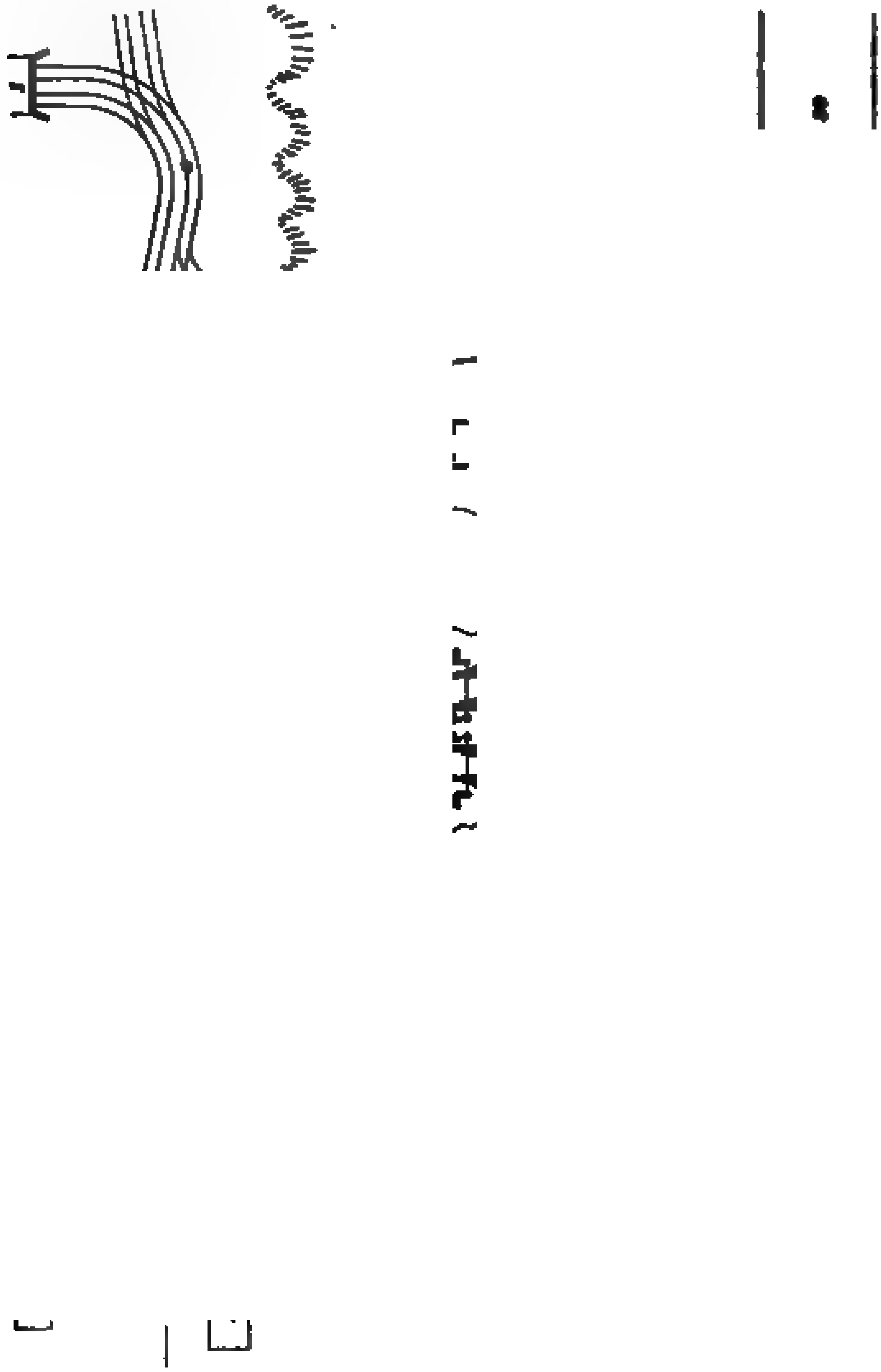
If the height or the distance is less than this, a trestle or embankment can be built out some distance towards the tipple, and the cars or coal can be lowered vertically to the tipple, as described farther on.

2631. A general arrangement of a mine opened with a gravity plane is shown in Fig. 965. Inclined planes may be of single track or of 3 rails, with passing points half way up. In Fig. 965 a double-track plane is shown branching into two outside empty tracks *b*, and a middle track for loaded cars *a* at the head. At the foot of the plane, the loaded cars land on the outside tracks *a*, and the empty cars are taken up from the middle track *b*.

From one to six cars are usually lowered at a time. More can be lowered. This depends considerably upon the length and grade of the plane. Where planes are long and much time required to lower cars, more are put in the trip than when short planes are used.

The arrangement of the tracks at the top is for haulage by a compressed-air or electric locomotive, or if a long haul

exists from the top of the plane to the mine mouth a steam-locomotive can be used between those points. If rope haul-



age is desired, the track arrangement of the top can be made similar to that at a drift mine. The empty return

track should be planned on the side convenient to the shops and buildings. A turnout can be planned as shown for running out loaded cars from the loaded track to provide coal to the boilers, and for running cars of rock to a waste dump.

Tracks should be arranged conveniently to the storerooms, material yards, etc., for running cars thereto that have been loaded at the foot of the plane and hoisted to the top. At the foot of the plane plenty of track room should also be provided for the accumulation of loaded and empty cars. This is of advantage in case of accident on the plane, as loading can continue with the coal that has collected, and many little delays may be thus overcome that would otherwise interfere with a large output.

If the conditions are favorable to planning the incline so that the loaded cars will automatically be discharged at the tipple without disconnecting the rope, it is of advantage, unless the conditions require the employment of many hands at the foot of the plane.

The arrangement of the buildings around the mine opening should be similar to that of a drift mine, except that the tipple platform and tracks should be removed to the foot of the plane, at which point the weighing is also generally done. If the ground is steep, the yard room will be narrow, and it will be necessary to locate the buildings in a line along the contour of the hill. The engine and boilers may be located on a lower level, and the ground somewhat excavated for the purpose, or they may be raised to the yard level with foundation walls. The boiler room should be located conveniently for the disposal of ashes.

If many supplies, lumber, and material are received by railroad, it may be necessary to provide a storehouse and yard room near the railroad, from which place they can be moved up the plane in empty cars as time will permit, unless it is done by teams over a wagon road.

Lumber, props, etc., may be cut on the property, or in the neighborhood, and be delivered by teams or tram-road to the mine mouth, in which case the lumber-yard will be located above.

2632. By referring to Fig. 965, it will be seen that coal from the drift *1* is taken over the track *a* through the wheel house *35*, under the drums *l*, when the proper rope is attached, and the cars run down the gravity plane on either track, hoisting the empty cars up the other. The drums are controlled by a brakeman in the shed shown at *34*. From the foot of the plane *p*, the loaded cars are run to the tipple *3*, passing over the scales *2*, by means of track *a*. The weighing is done by the weighman in his office *32*. The empty cars are returned to the drift mouth over tracks *b*. Track *h*, at the head of the plane, is intended for the accumulation of about two trips of loaded cars. The haulage is accomplished by a compressed-air or electric locomotive, which is housed in building *37*. Sawed lumber from the yard *17* adjoining the sawmill *18* is transported to any part of the plant over track *d*. Track *c*, with its branches, reaches the carpenter shop *16*, the blacksmith shop with tool-rack annex *15*, the machine-shop *14*, locomotive house *37*, supply house *20'*, with supply-clerk's office *21*, the iron and pipe shed *23*, and oil house *22*. Rock cars are taken to the rock dump over tracks *c* and *o*. Coal for the boilers is run over track *e*, which is continued on to track *o* for the removal of ashes.

On the platform at the head of the plane, at the points *H*, *H*, are locks actuated by levers to prevent cars running to the head of the plane. A plan of one of these locks is shown on the right of the plane. On the plane at *I* are safety-switches controlled by the brakeman in shed *34*. A plan of these switches is shown on the left of the plane. In the event of a runaway car, the brakeman throws a convenient lever, which removes the tension from the rope *s*, and allows the weight *r* to throw the switch so that the runaway car will be thrown off the plane. At the foot of the plane are located a small repair-shop *33*, to which crippled cars are run over track *g*, a receiving house for supplies *20*, and a timber and rail yard *19*. The supplies, timber, and rails are received from railroad-cars on track *F*, and are sent to any part of the plant over mine-car track *f*.

The fan is shown at *7*, with an underground connection with the traveling and air way *36*. The tank for water-supply is shown at *10*, with supply pipes ——·——·—— running to various points. The wash-house is shown at *11*, the mine office at *12*, and the lamp house at *13*. The compressed-air or electric plant is shown at *8*. The stable *25*, wagon and harness house *26*, and the hay and feed storehouse *27* are shown in the upper left corner of the plan. The powder house is not shown. It may be located in any safe place, 1,000 feet or more from the other buildings. The tracks shown by dotted lines near the drift mouth are intended to show connections with other distant productive openings. In case the level ground from the foot of the plane to the railroad-tracks is not limited, the tracks to the tipple can be continued straight from the foot of the plane, instead of deflected as shown on plan. The location for a coke-oven plant is shown at *30*. The shipping tracks are similar to those shown in previous plans: *A* is the empty shifting track, *B* the lump-coal track, *C* the nut-coal track, and *D* the slack track.

2633. A circular saw will not be needed if timber is delivered ready cut. If lumber is to be cut from standing timber, it may be necessary to erect the sawmill at some distance from the mine. In some mines opened with gravity planes, the requirements for the operations are so few and simple that most of the buildings are located at the foot of the plane, a blacksmith shop and a few supplies being located at the top. The mule stables are sometimes located at the foot of the plane.

In the event of all the buildings being erected below, and it is desired to erect a rope-haulage plant, it may be placed at the foot of the plane, and the ropes be led up to and into the mine opening. If the haulage is to be done by locomotives, and there is other machinery needing attention and repairs, the surface buildings are more conveniently arranged around the top of the plane.

2634. Where the hillside is very steep, the cars of coal can not be run on the inclined track of the plane, as some of the coal would spill out of the car. It depends upon how full the cars are loaded as to what will be the maximum inclination they will run on without spilling. Cars loaded above the top should not be run on an inclination of much over 15°. Cars loaded level full may be run on an inclination of 20° for short distances, but for these and steeper

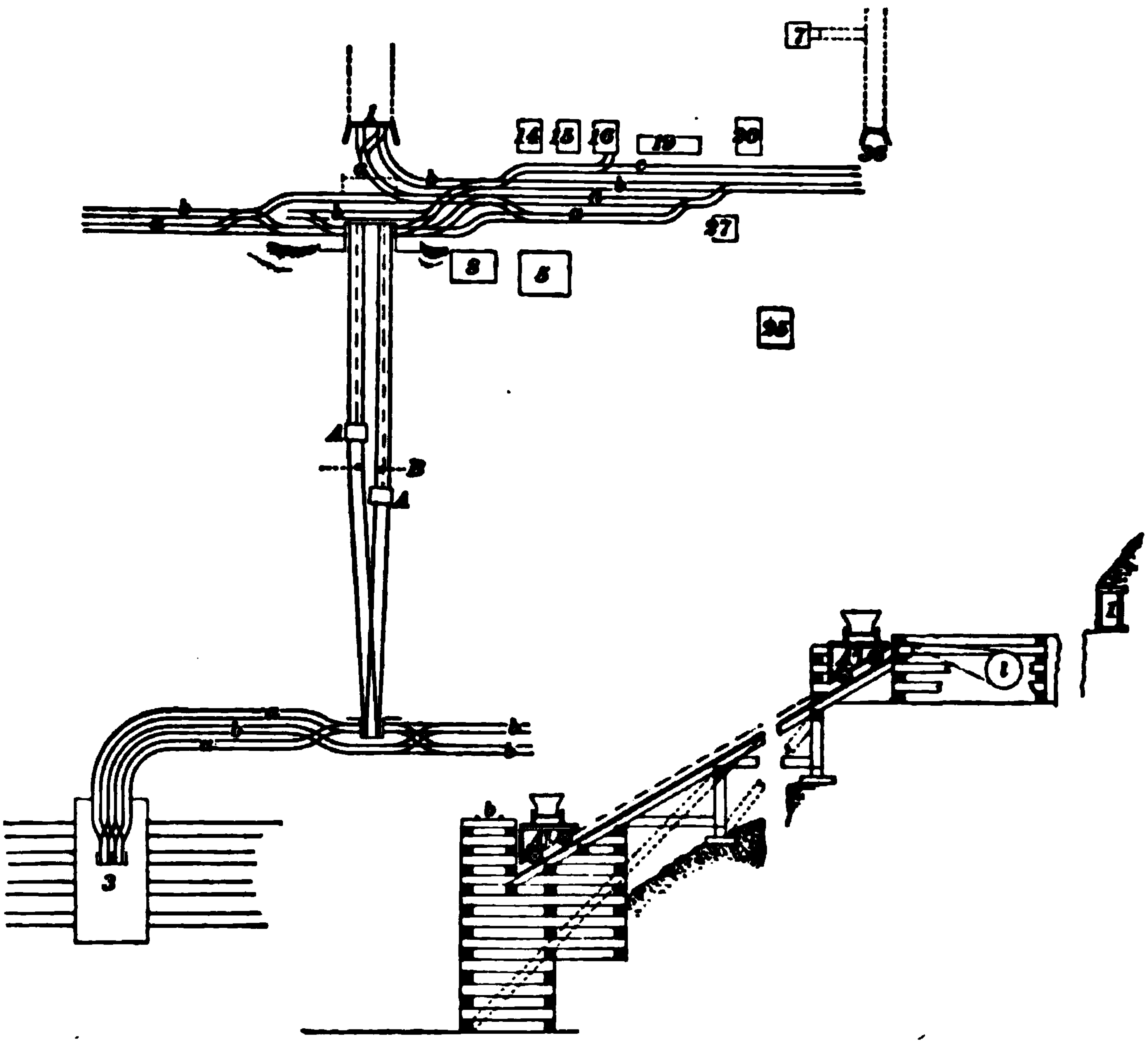


Fig. 966.

grades a device known as a **dummy,** or platform car, is used, on which the cars are lowered. This is shown in plan and profile in Fig. 966. In consists of an inclined truck *A*, built on the same angle as the plane. The car rests level on this in its movement on the plane. The arrangement of the tracks for handling the loaded and empty cars on and off the dummy is shown in the same figure. At the top it is

necessary to first remove the empty car before running the loaded car on. The loading at each side of the plane is done at separate opposite landings.

The dummy tracks are made to converge after passing the mid-point of the plane *B*, so that at the foot of the plane they have the same landing, and the empty cars are pushed on the dummy as the loaded cars are pushed off. This is done by having the two inclined tracks come together at the foot of the incline without passing switch rails, but so that their centers are about 4 inches apart. This necessitates that the platform of one dummy extend 4 inches farther towards the landing on the loaded side below than that of the other dummy. The outside rails of the dummy tracks must be slightly elevated after passing the mid-point of the plane.

If there is sufficient room above and below, the mine-cars can be run on and off the dummy in the same direction as the plane, that is, by tracks at right angles to those shown in Fig. 966, so that the amount of back-switching will not be so great above.

It will not be possible in this case to remove the loaded cars at the foot of the plane from one side and place the empty cars on from the other.

2635. In the plan shown in Fig. 966, the mouth of the mine is shown at *1*, with tracks for loaded cars *a* and empty cars *b* leading to and from the head of the plane. The drums, shown in profile at *l*, are not shown on the plan, as they are under the platform. The fan *7* is shown in connection with the airway and traveling way *36*. On the plane the dummies, or platform trucks, are shown at *A*, *A*, and the middle point of the plane where the tracks begin to converge is shown at *B*. The tipple *3* is shown to the left of the foot of the plane, with loaded tracks *a* and empty tracks *b*. The air-compressing or electric plant is shown at *8*, with the boilers *5* adjoining. The stable *25* and feed house *27* are shown on the same side of the tracks as the boilers. On the other side of the tracks are shown the machine-shop *14*, blacksmith shop *15*, carpenter shop *16*, material yard *19*, and

supply house *20*, all reached by mine-car track *c*. In the profile the drift mouth is shown at *1*, the drums, under platform, at *l*, one dummy *A* at the top landing and the other in the pit at the bottom.

2636. In Fig. 967 is shown a device, known as a **barney,** for lowering loaded and raising empty cars without attaching a rope to the cars.

The barney *A* is a truck running on a narrow track between each of the inclined tracks. A large timber forms the body of this truck, and the ends of the ropes are attached to a barney on each track. A loaded car is pushed

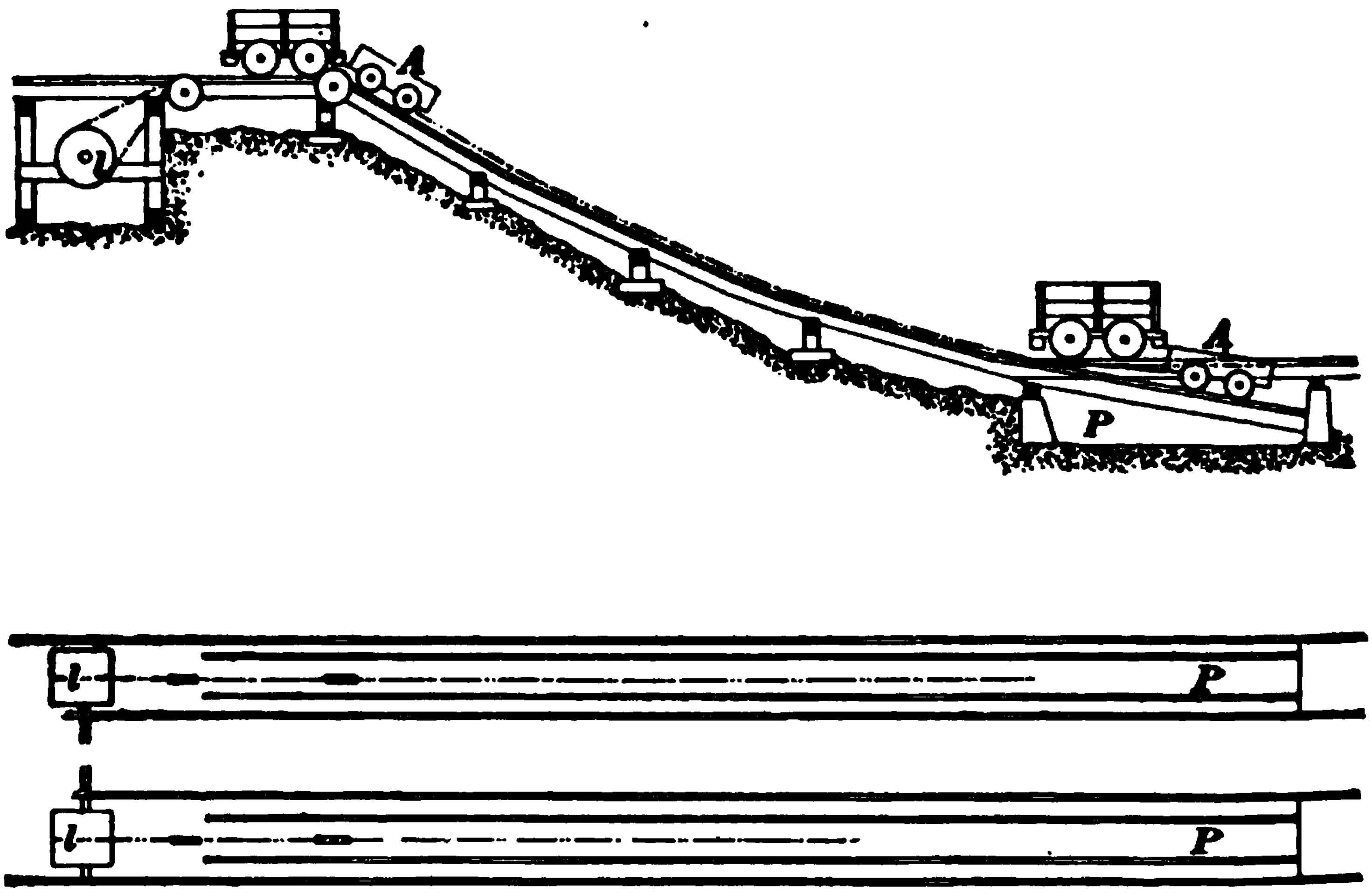


FIG. 967.

against the barney at the top, which has landed at the knuckle, and the weight of the car against the barney causes both to move down the plane, and the speed is regulated by the brake on the drums *l*. As they descend, a car is moved up the plane on the other track with the barney behind it.

When the car and barney reach the bottom of the plane, the barney goes beneath the level of the other track into a pit *P*, allowing the loaded car to pass over it. The barney remains in the pit to be drawn up behind the next empty

car when it is pushed in front of it. This arrangement avoids delay in coupling and uncoupling the rope from the car.

If the mine-car must be moved some distance after reaching the bottom of the plane, this arrangement is of advantage, as with its momentum, after being released by the barney, it will run several hundred feet on a level. In this arrangement a counterweight should be caught up by the barney landing at the top of the plane, to check its momentum.

2637. If it is desired to lay a 3-rail inclined plane, on account of length or the cost of construction of wide embankments, or on account of the arrangement of landings at the top or bottom, such a plan can be used as is shown in Fig. 968, which also shows a system of tracks at the bottom, where the cars arrive and depart in trips. This operates as follows:

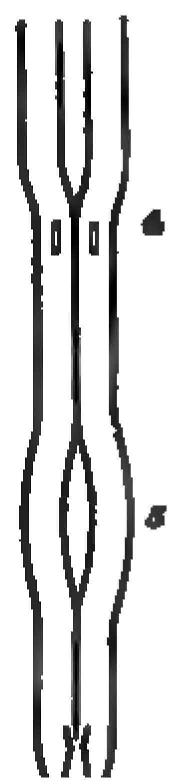


FIG. 968.

When the weight on the switch-stand *W* is placed at *A*, the loaded cars are lowered to the siding *C*, and the empty cars are taken up the plane from the siding *F*, while a trip of empty cars is coming in at the siding *E*.

When the weight is at *B* on the switch-stand, the loaded

cars are lowered to the siding *D* and empty cars are taken up from the siding *E*, while a trip of empty cars is coming into the siding *F*. In Fig. 968 the knuckle at the head of the plane is shown at *4*, the mid-distance and divergence of rails into double track is shown at *5*, the scales are shown at *2*, and the tipple at *3*.

LENGTH AND GRADES FOR INCLINED PLANES.

2638. For lengths up to 500 ft. the grades should not be less than 5% for a weight of 8,000 lb. in the descending and 2,800 lb. in the ascending load, or 5½% for 4,000 lb. descending and 1,400 lb. ascending load.

For planes from 500 to 2,000 ft. the grades should be from 5% to 10%, depending upon the loads. For a plane 2,000 ft. long and 10% grade, 4,000 lb. descending load will hoist a 1,400 lb. empty car.

There are instances in which 25 and 30 loaded cars will descend a 1¾% grade, but where planes are lighter than 5% grade the empty cars will usually have to be hoisted on a plane operated by an engine, in which case the tracks can be single or double.

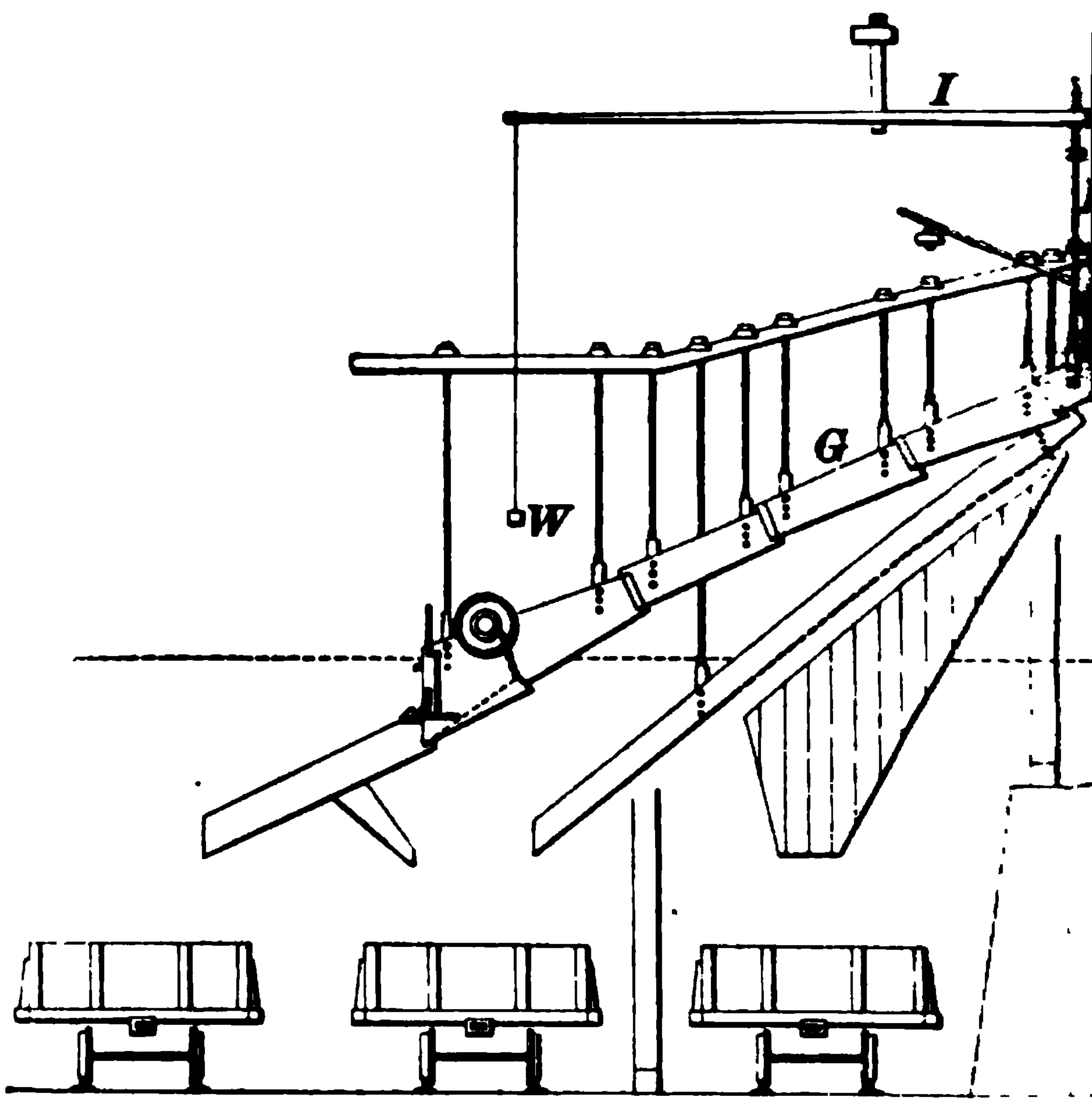
An inclined plane should be slightly concave in profile, and steeper at the top than at the bottom, except where a dummy is used, which necessitates that the grade be continuously the same on account of the dummy being built only for one angle of inclination, and it therefore can not be operated on a plane with much variation in inclination.

COAL-HANDLING APPLIANCES WHERE PLANES CAN NOT BE USED.

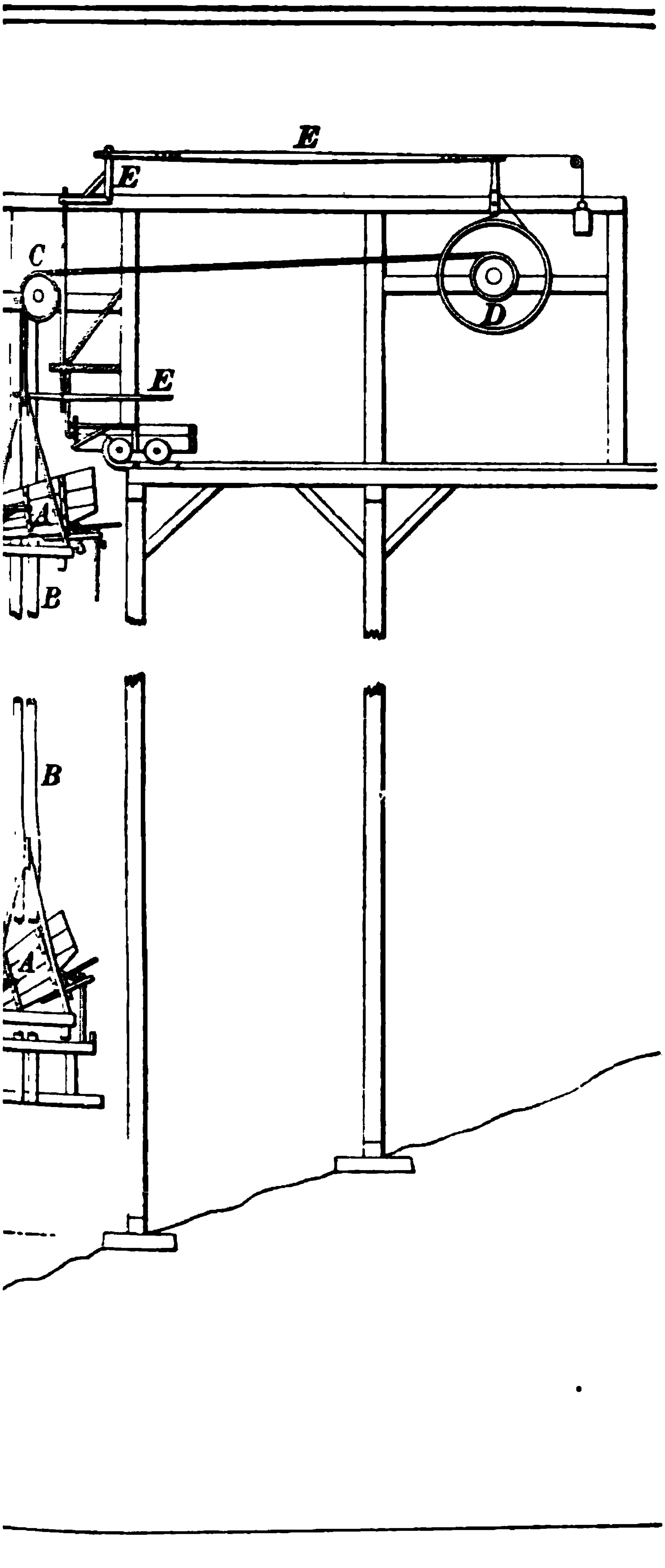
2639. The height for a tipple above the railroad-tracks has been given at 20 to 35 feet, although it may be a few feet higher if the coal is hard enough and the screens are suitably arranged.

Where the coal outcrops on the hillside at a greater elevation than this and up to 45 or 65 feet above the railroad-

F



FIG


E
E
C
D
E
A
B
B
A

tracks, the mine should be opened, if possible, at some distance from the location for the tipple, so that the cars can be brought from the mine outlet over a tram-road 500 to 1,500 feet long, with a 1% or 2% down grade to the tipple landing.

If this plan is not feasible, and if, as mentioned under Gravity Planes, the distance, the height, and the length of plane are not sufficient to warrant the construction of the latter, then the following devices should be used for handling the coal to the tipple :

2640. Fig. 969 shows an arrangement suitable in case either the elevation is insufficient to warrant an inclined plane, or the hillside is too steep to permit of its construction, even if there is sufficient height. This consists of one or two baskets *A*, operating vertically in guides *B* between the upper and the lower landing. The arrangement is similar to that of a shaft hoist, but simpler, as the descending load furnishes the operating power.

The coal is dumped into the basket from the mine-car at the upper landing. This weight then causes the descent of the basket and load, which is suspended by a rope led by an overhead pulley *C* to a drum *D*, with a brake *E* for controlling the descent. The descent of the basket and load raises either a counterweight or a second empty basket. When the basket reaches the lower landing, which corresponds to the tipple landing, a front gate to the basket *F* is opened, and the load is dumped automatically into a chute *G* provided with screens. The basket then rights itself as the brake is released, and the empty basket is caused to ascend by the descent of the counterweight, or a second basket with its load. This device requires few hands at the lower landing, and is suitable where the coal is not seriously injured in dumping.

The automatic device for opening the front gate of the basket consists of a hooked bar *H* attached to a pivoted bar *I*, which is balanced and held in position by the weight *W*.

If the coal is soft, instead of dumping into a basket at the tipple landing, the car of coal should be lowered by a device, shown in Fig. 970, to the tracks of the tipple landing, and from there run to the chutes for dumping.

2641. In Fig. 970 the loaded cars are run onto the cage A, and the empty cars at the tipple level are run onto

FIG. 970.

the cage B. By means of the brake drum D, the loaded cage A is lowered down one side of the vertical hoist C, and its weight raises cage B to the top.

These devices are best adapted where the height of the upper landing is at least 20 to 30 feet above the tipple landing, and may be used for greater height; although, if the elevation much exceeds 50 or 60 feet above the lower landing, or 70 to 85 feet above the railroad-tracks, even on a steep hillside, it will be found preferable to introduce an inclined plane operated with a dummy.

ARRANGEMENT OF TIPPLE STRUCTURE AND FITTINGS.

OPERATIONS AT THE TIPPLE.

2642. The cars of coal from either a shaft, slope, drift, or gravity-plane mine, after being run over tracks more or less long, are delivered at a tipple, where the coal is dumped into a chute, with or without screen bars, or into a bin. The loading into railroad-cars or other means of transportation may be controlled by gates in the chute, or baskets, which check the force of the coal as it descends the chute, and lower it gently into the cars, thus reducing breakage.

The weighing may be done either before the mine-car is dumped on the platform or after it has been screened, and the lump coal held in a basket connected with scales; or it may be weighed in the railroad-cars.

It may also be necessary to provide facilities in the tipple to clean the coal by picking out the slate from the lump coal or screenings, or both.

If there are cars of rock from the mine to be unloaded outside, tracks may be necessary at the tipple to run these cars to waste dumps, or else to a chute for loading into railroad-cars, to be carried away as waste.

2643. The arrangement of the fittings and devices in the tipple structure can best be described under the following operations at the tipple:

1. Dumping coal from the mine-cars.
2. Chuting the coal.
3. Screening coal.
4. Loading into railroad-cars.
5. Weighing.
6. Cleaning coal.
7. Handling of rock.

DUMPING.

2644. Dumping may be done in tipples of three classes, viz., in push-back tips, in revolving or oscillating cradles, or in cross-over tips.

PUSH-BACK TIPS.

2645. Fig. 971 shows an ordinary form of tipple in which the car of coal is run to the tipple horns, and after being dumped it is pushed back to a switch before the next load can be moved to the tipple. The axle *A* on which the tipple horns *B*, *B* turn is slightly back of the center of gravity of the loaded car, so that the force with which the

Fig. 971.

load dumps is sufficient to raise a counterweight *W* at the rear of the tipple. As this raises and the car dumps, a brake *C* is applied by foot or hand to hold the counterweight and tipple in position until the car is empty.

The brake is then released, and as the position of the center of gravity of the empty car and raised counterweight is then back of the tipple axle, the weight of the counterbalance returns the empty car and tipple horns to a level position.

2646. Fig. 972 shows a similar tipple provided with buffer springs *S*, *S*, which take up the jar caused by the wheels of the loaded car striking the horns *B*, *B*. This is known as the Phillips automatic push-back car tip. The recoil of the springs checking the loaded cars serves to push back the empty car, after it is dumped, through the medium

of the arms C, C, against which the wheels strike. Sometimes the axle on which the tipple turns is elevated so as to be at a point a little lower than the center of gravity of the

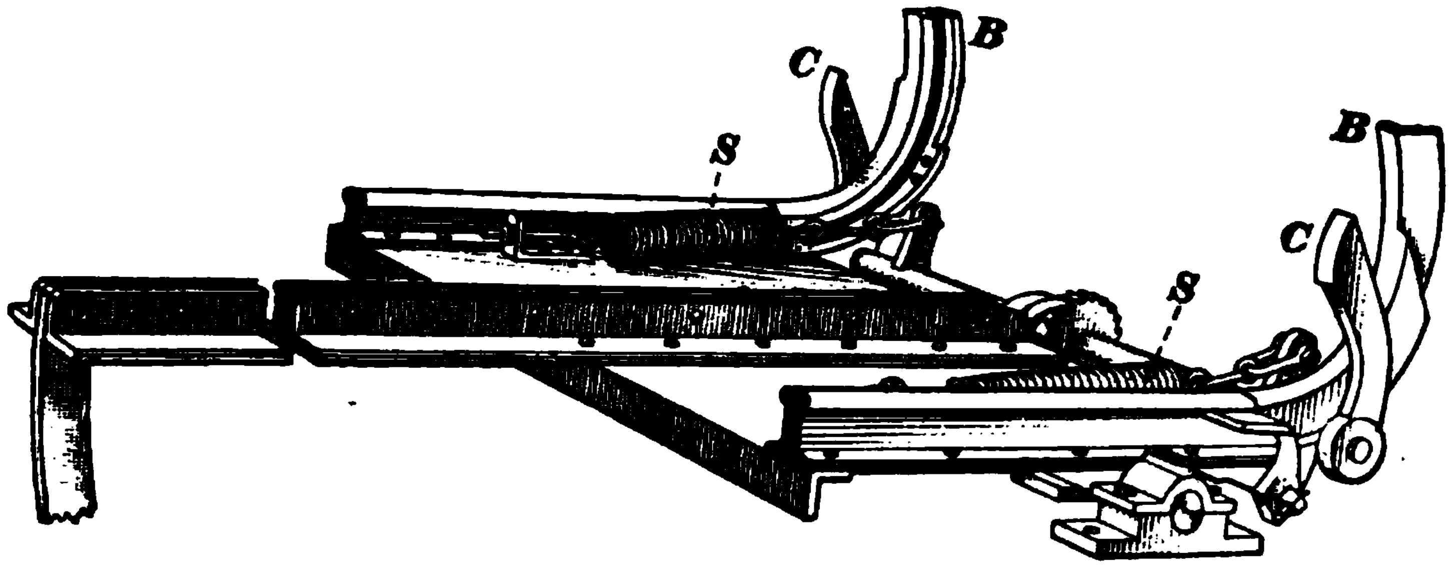


FIG. 972.

car when loaded, and a little above it when dumped. This does not necessitate the use of counterweights.

REVOLVING OR OSCILLATING TIPS, OR CRADLES.

2647. These may make a full revolution or part of a revolution and return. They may turn either in the direction of the tipple or at right angles to it. Fig. 973 shows a usual form of this class. The door A, shown on the top of the tip, or cradle, is for a special case in releasing very soft coal to reduce breakage in handling. By referring to Fig. 973, it will be seen that the mine-car is run into a box-like arrangement which revolves on the axles C, so set that the center of gravity of the loaded car is above the axles. The overturning of the box is regulated by the brake-wheel B, assisted by the counterweight W. When the coal has been dumped into the chute D and the brake-wheel has been released, the dump rights itself automatically. Cars dumped in this style of cradle do not need end gates.

The system of tracks where tipples like those already mentioned are used may be similar to any of the arrangements already shown for shaft, slope, drift, or gravity-plane mines. These plans are interchangeable for mines opened either way, according to the conditions and requirements.

As time is required in pushing back an empty car from the tipple before another loaded car can be pushed there, it

is necessary, if speed is desired, to provide two tipple horns, fed by one or two loaded tracks, and an empty track, so

Fig. 973.

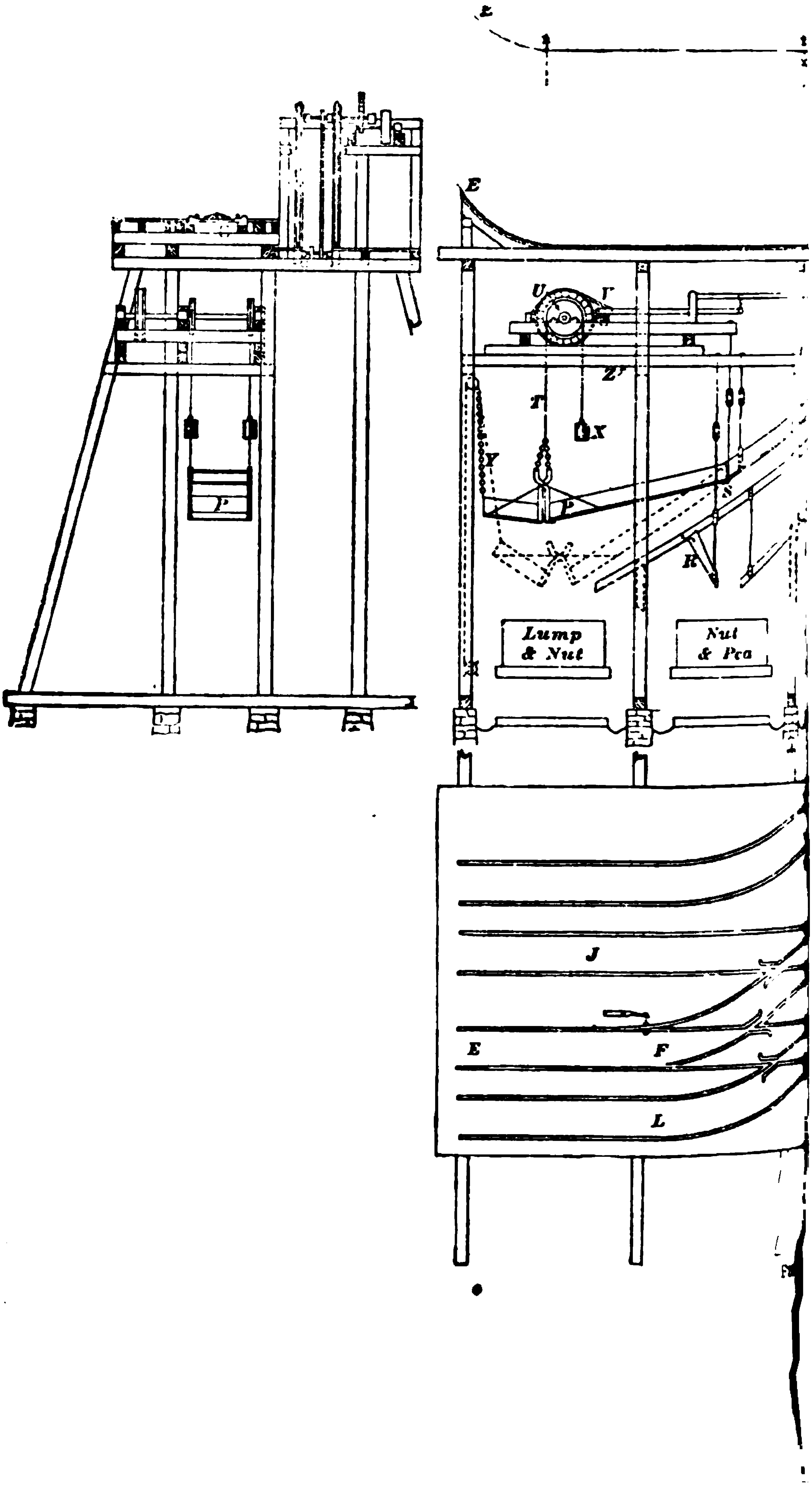
that the empty car can be quickly moved onto it after dumping, to make room for the next loaded car.

CROSS-OVER TIPS.

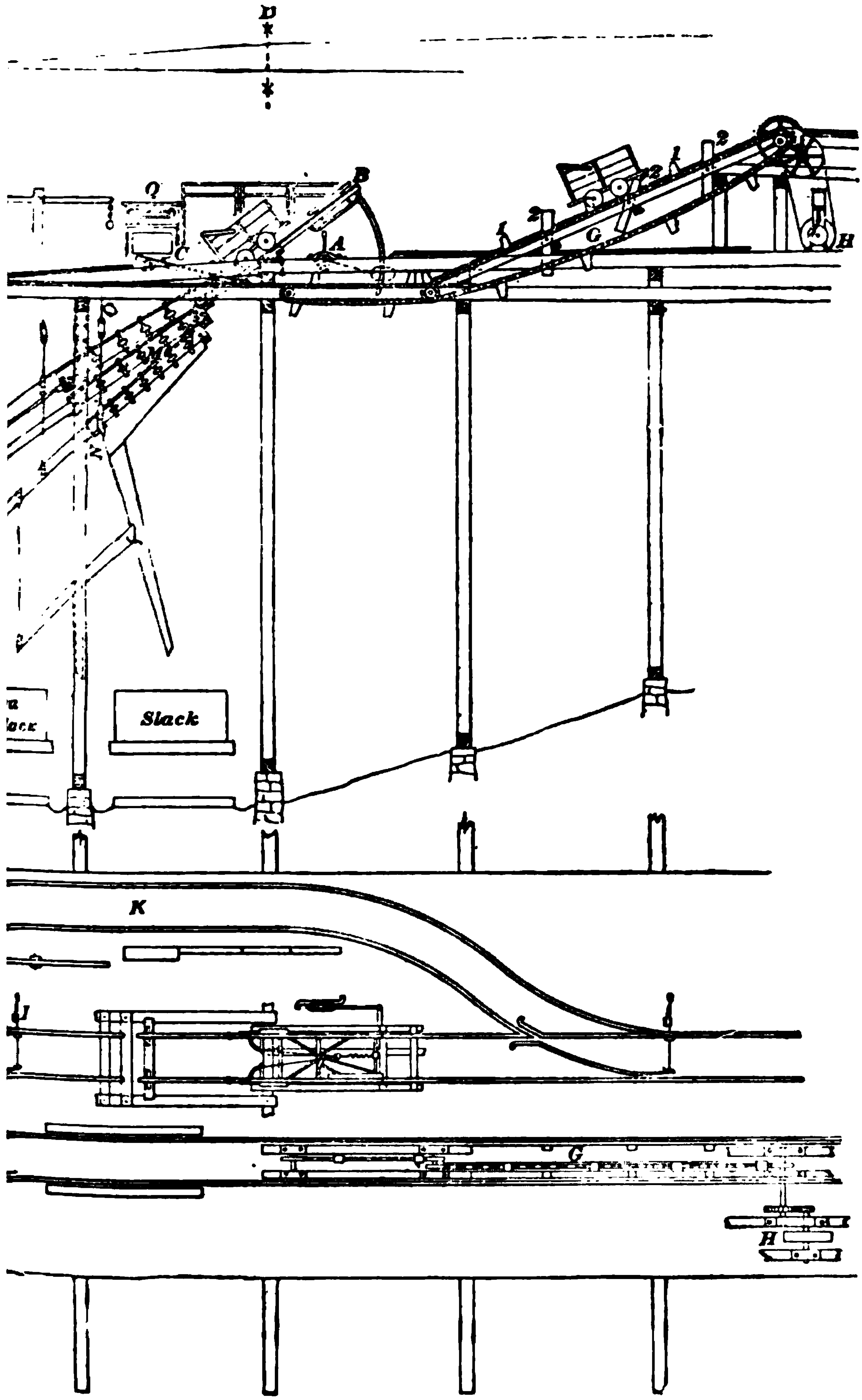
2648. In tipples of this class, the cars of coal after being dumped pass beyond the tipple over rails spanning the dump hole to an empty track and return to the mine by a switchback, or by making a full or half circuit.

The arrangement of the tracks for these tipples is different from that of the push-back or the revolving type. Generally, one tipple of this kind is sufficient, requiring but one loaded and one empty track.

2649. In Fig. 974 is shown an arrangement of tracks for cross-over tips. The profile at the top shows that the loaded cars approach the dump *D* by gravity over a 1.66% grade, and the empty cars leave the dump on a 6.25% grade for a short distance. They then run to the end of the tipple



E
U
V
Z'
T
X
Y
P
P
R
Lump
& Nut
Nut
& Pea
J
E
F
L


D
Q
B
C
A
G
H
K
Slack
J
G
H

platform over a level track, till, near the end, they run part way up a short incline *E*. Running back down this, they cross the spring switch *F*, shown on plan, and run down a 1% grade to the foot of an endless-chain hoist *G*, which is run by the engine *H*. This hoist raises the empty cars to a sufficient height to allow them to run by gravity to the collecting point for empty trips. Crippled cars are passed by means of switch *I* to track *J*. Rock cars are run to a dump over track *K* and are returned from the dump over track *L*. The sprocket-chain used on the hoist *G* is supported on the plane by a bar of flat iron, over which it slides. The lugs *1* on the sprocket-chain move the car by engaging with a block bolted to the car, back of the axle. The stops *2* are pressed down by the bumpers of the car as it moves up the plane. When released, they prevent cars running down the plane.

2650. The principal tipples of this class are the Mitchell, the Wilson, and the Phillips. In these the loaded car comes to the tipple by gravity and is stopped by the horns of the car tip. Buffer springs may be provided at the horns or elsewhere to take up the jar caused by the speed of the loaded car. The tip brake *A* is then released and the contents of the car is dumped into the chute. A counterweight is raised as in ordinary tipples, the brake is applied, holding it up and the car in position until the car is empty. The brake is then released, and the car drops back to a level position while controlled by the brake.

The empty car remains in this position while a loaded car moves towards it. The wheels of the loaded car depress a tread rail *B*, which operates a system of levers, causing the tipple horns to spread out, or turn down and outwards, so as to clear and release the empty car, which runs by gravity off of the inclined track of the tipple and passes the horns before the rear wheels of the loaded car leave the tread rails of the horn spreader.

As the rear wheels of the loaded car clear the tread rail, springs cause the horns to come back to their first position to check the loaded car.

The empty car crosses the dump hole on hinged rails *C*, attached to the end of level rails at the tipple horns. The other ends of the hinged rails are free to slide when the car is being dumped, but making a butt joint with the permanent track beyond the dump hole for the passage of the empty car when released from the tipple. The rails are about 8 feet long.

The length of the tipple rails is about 11 feet, and the rail spreader about 4 feet long, though this length will vary, depending upon the length of the car.

The tread rail and horn spreaders, with their levers, are connected with the rails of the tipple, and raise with them when the car is being dumped.

2651. The Mitchell tipple is shown in outline in Fig. 974. The horns spread out by the passing of the mine-car over the tread rail. If cars are very wide, the hinge rails, instead of being attached to the end of the level rails of the tipple, are caused to spread outwards by levers and clear the width of the car.

2652. The Wilson tipple is shown in Fig. 975. In this

Fig. 975.

tipple the spreading of the horns *B*, *B* is accomplished somewhat differently. Here the bumpers and bottom of the car depress a lever *A* connected with the horn spreaders. The approaching loads are checked by a second set of

smaller horns *C*, *C* back of the tipple, opened by a lever *D*, and held open by the car passing over a tread rail long enough to permit the passage of the car before they close.

2653. Fig. 976 shows the Phillips automatic cross-over tip. In this tip the horns *A*, *A* roll outwardly, releasing the empty car by the passing of the loaded car over the tread rail, which operates the horns by a chain *B* working a system of levers.

The horns are pivoted in wrought journal-boxes. The axles *C* on which the horns roll out extend some distance

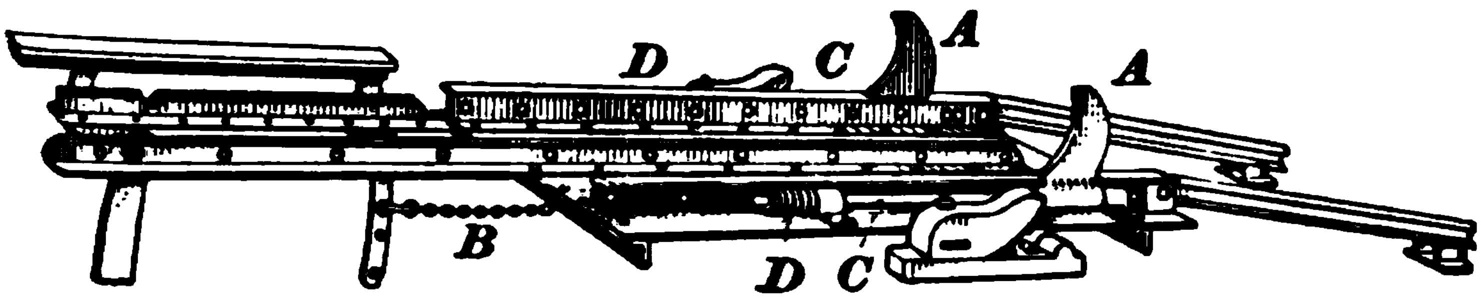


FIG. 976.

back along the outside of the rails of the tipple, and their rear ends are provided with special springs *D* to take up the jar caused by the loaded car striking the horns.

CHUTING.

2654. The coal is dumped into a chute which is made either short or long, as the tipple is at a small or a considerable height above the loading point.

Chutes are from 4 to 5 feet wide, with sideboards 1 to 3 feet high, supported between the posts of the bents forming the trestle supporting the tipple structure and platform. The chute bottoms are covered with $\frac{3}{16}$ inch to $\frac{3}{8}$ inch sheet iron or steel.

The angle of the chute required for the proper movement of the coal upon an iron or steel surface will vary from 26° to 29°, depending upon whether the coal is in lumps and dry, or whether the mine-run coal contains much fine coal. For small screenings the angle will vary from 30° to 33°, and possibly more if they are wet.

It is desirable to have the chute rigged to a windlass for regulating its angle, if there is much variation in these respects.

The chute at the loading point should be 2 or 3 feet above the railroad-car. By drawing a line from this point on the angle of the chute (29°), its intersection with elevation of the tipple platform will determine the location for the tipple. The center of the car tip should be located from 3 to 5 feet in advance of this point, so that when the car is dumped there will be a drop of 1 to 2 feet from the car to the chute, so that all the coal will run rapidly out of the car.

A chute may be of very simple construction and have no provisions for screenings, weighing, or regulating the loading, unless it be an iron apron, or gates, to hold the coal back while shifting railroad-cars into position for loading.

If only screening is necessary, screen bars may introduced in a chute of wooden frame. If special arrangements for weighing and loading are necessary, a chute of all iron or steel is preferable, although in some cases part iron and part wood chutes are used.

SCREENING.

2655. The usual sizes of coal produced are known as lump, nut, pea, and slack. Although nut coal has been known to be from $\frac{7}{8}$ inch to $2\frac{1}{2}$ inches in size, and pea coal from $\frac{1}{4}$ inch to $\frac{5}{8}$ inch, the following sizes are now the standard:

Lump.—All coal passing over $1\frac{1}{2}$-inch screens.

Nut.—All coal passing through $1\frac{1}{2}$-inch openings and over $\frac{3}{4}$-inch openings.

Pea.—All coal passing through $\frac{3}{4}$-inch openings and over $\frac{3}{8}$-inch openings.

Slack.—All coal passing through $\frac{3}{8}$-inch openings.

2656. Long and flat coal may pass into these sizes that will be larger than the sizes indicated, and some small coal may go with the larger coal when the coal has not been thoroughly screened.

The screening may be done over (1) flat bar screens, (2) over revolving or shaking screens.

The former are generally used, and the coal is screened as it moves by gravity over the bars.

The latter methods require power to operate the screens for the movement of the coal over their surface. If, however, there is not sufficient elevation between the railroad-tracks and the tipple platform, or if the amount of screenings is very large and perfect screening is a matter of importance, their use is necessary.

2657. The standard sections and spacing of flat screen bars are shown in Fig. 977. *A* shows diamond-shaped lump-coal screen bars, and *B* shows another form of lump-coal

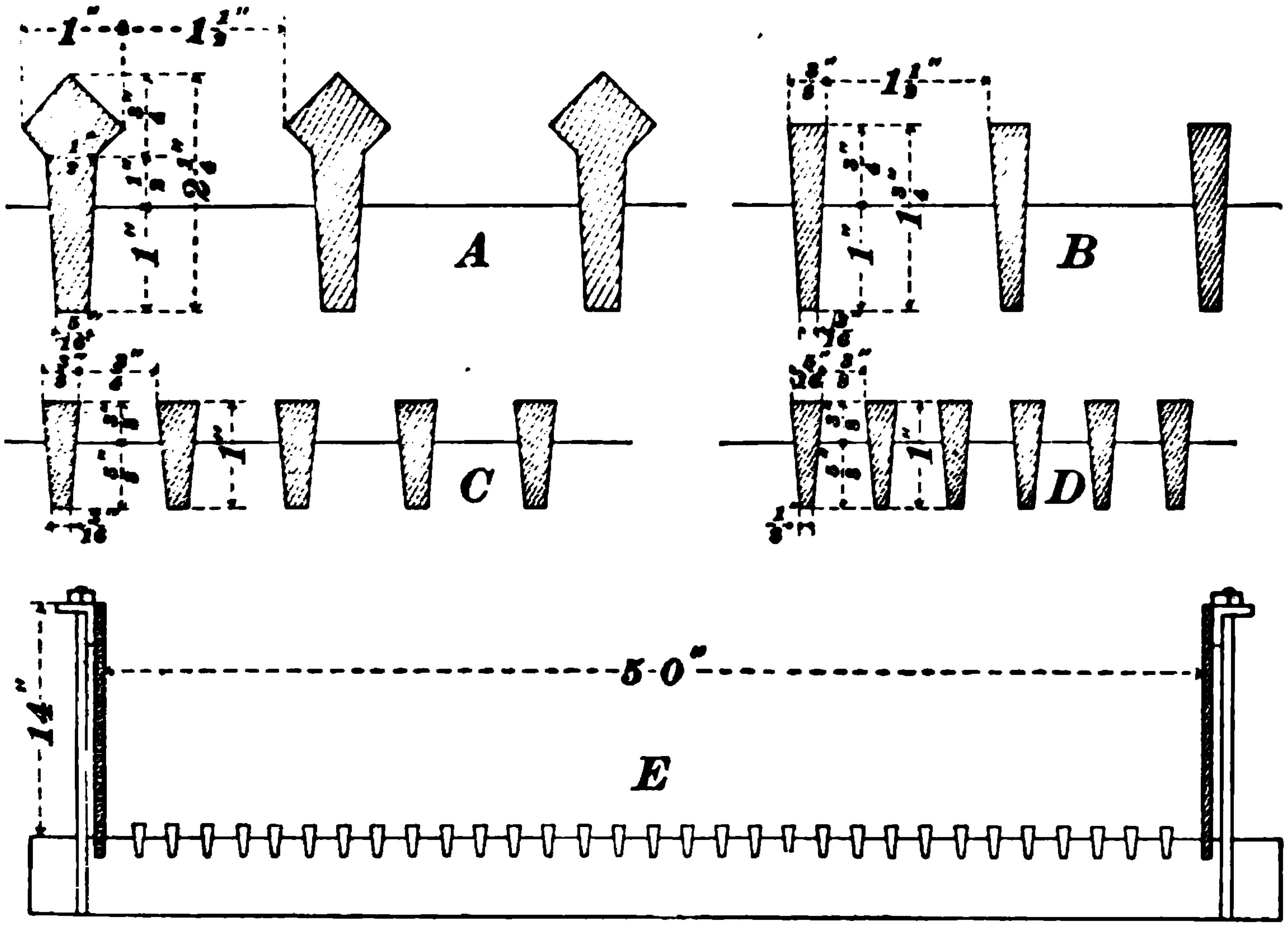


FIG. 977.

screen bars. *C* shows nut-coal screen bars and *D* pea-coal screen bars. *E* shows a cross-section of a lump-coal screen. For lump-coal screens, the screen bearing bars are ¾ inch thick by 4 inches deep, and for nut and pea coal the screen bearing bars are ⅝ inch thick by 3½ inches deep.

These bars are spaced in standard screen frames or chutes 5 feet wide and 12 feet long, with steel sides 8 to 16 inches high. The frames for the smaller screenings are sometimes shorter than this.

Screen bearing bars ¾″ × 4″ and ⅝″ × 3½″, notched to receive the bars, run across the under side of the frame about 2 feet

apart; their outer ends are held by a stirrup on the lower end of a bolt which pierces the upper flange of the sides of the chute frame, where it is secured by a nut drawing the bearing bars up tight against the frame.

The screen frames have a sheet of steel about ¼ inch thick, 1 foot wide, running across the ends of the frame. Below these are cross timbers to which the plates are bolted, giving rigidity to the frame.

The ends of the screen bars are cut down so that at their upper and lower ends they lap under these end steel plates, flush with their surface, which prevents their rising out of their bearings.

2658. The arrangement of the screens, generally one below the other, is shown in Fig. 974 at *M*. There will be 1 to 3 screen frames, according as 2 to 4 sizes are to be produced.

The lump screen, at the upper end, has steel plate for a length of 4 feet, which receives the blow of the coal as it is dumped, allowing it to spread before it passes over the 12-foot length of screen bars.

The screens *M* are supported by rods *N* to the upper platform, provided with turnbuckles *O* for adjusting the angle of the chute, which may vary slightly with different coals.

To the end of the screen frames are joined steel chute frames of the same width as the screens, but with sheet-steel bottoms to lead the coal to the points where it is to be loaded on railroad-cars, or to loading baskets *P*.

The chutes are provided with fly doors *R*, or gates, by which the sizes can be turned from one loading point to another, and produce either various mixtures or separations of the sizes, according to requirements.

LOADING.

2659. The coal from the screens and chutes may be either run into bins over the railroad-tracks and held there for loading, as needed, or it may be loaded directly into railroad-cars, its flow being regulated or checked by gates, loading baskets, etc.

Generally, where coal is shipped, the storage capacity of the bins is not large, and serves only for holding screenings, in case of short delays during the day's work, as in the supply of railroad-cars, or while shifting them.

If coal is to be stored for many days, an extended arrangement of bins, similar to those in use at coke-ovens, is necessary, with tracks and switches leading thereto from the tipple, or else it is stored by means of a system of elevators and conveyors removing the coal from the tipple.

The point where the coal discharges from the chute into the car should be so arranged that the coal will be loaded evenly in the car on both sides of the center, so as to require a minimum amount of labor in trimming; and the end of the chute should be high enough above the car to clear the height to which the coal may be loaded above the sideboards of the car.

In loading ordinary coal-cars from a chute with end discharge when the apron is down, it should extend about 1 or 2 feet into the car and be about 3 feet above the sideboards, depending upon the height to which the car is loaded.

When the coal has been loaded to the proper height in one part of the railroad-car, the car is shifted and the loading begun at another point. Where only one chute is used, the car must be shifted twice as often as where two chutes and tipples are used. On this account, two chutes are generally used where the arrangements of dumping, screening, etc., are simple. This is also desirable in tipples fully equipped for automatic dumping, screening, and loading devices, if the track arrangement will permit and the cost can be incurred.

2660. In Fig. 974 is shown an arrangement for loading coal-cars by a center discharge basket *P*. This basket is made 12 to 24 feet long, depending upon the length of the chute and the height of the load in the railroad-cars. Its upper end *S* is hinged to the chute, and, when closed, rests at an angle of about 12° to 16°. This and the length of the chute are regulated to check the speed of the coal down the chute, so that it will land near the center of the basket.

The weight of the coal in the basket, which is suspended by a wire rope or chain *T* to a drum *U* controlled by a brake *V*, causes the basket to lower when the brake is released.

The descending load raises a counterweight *X*, and, as the load descends, the basket is opened by a chain *Y* adjusted so as to cause the outer arm of the basket to spread apart from the long arm, and open at a less or greater height above the bottom of the railroad-car as the height of the load increases.

Before the momentum of the descending basket and load has spent itself, as the coal discharges from the basket the brake *V* is applied to the drum above, holding the counterweight *X* in position to return the basket to its first position, when the brake is released.

The discharge angle of the basket, when open, will vary from 26° to 33°, depending considerably upon the nature of the coal and whether mine-run with much screenings is at times to be loaded on the lump-coal track or not.

2661. There are numerous arrangements of coal tipples, planned to suit the various requirements as to size of screenings to be produced and whether these are to be loaded on cars on one or more tracks, and whether mixtures are to be produced, etc. One other form of tipple, shown in Fig. 978, will suffice with that shown in Fig. 974 to illustrate the different principles. In Fig. 978 is shown a steel-built tipple, with screens, for producing three sizes of coal. This can also be arranged for producing four sizes. The sizes can be loaded separately or mixed, as shown.

A special feature is in the provision for loading box or gondola coal-cars on the outer track with either lump, lump and nut, or run-of-mine coal.

A receiving chute *1*, with two point chutes *2*, *2'*, is suspended beneath the weigh basket *P*. The upper end of the chute is pivoted to the steel structure by rods and turnbuckles. The lower end is provided with a windlass, so that the pitch of the chute can be readily changed to accommodate high or low cars. The chute is provided with a hinged

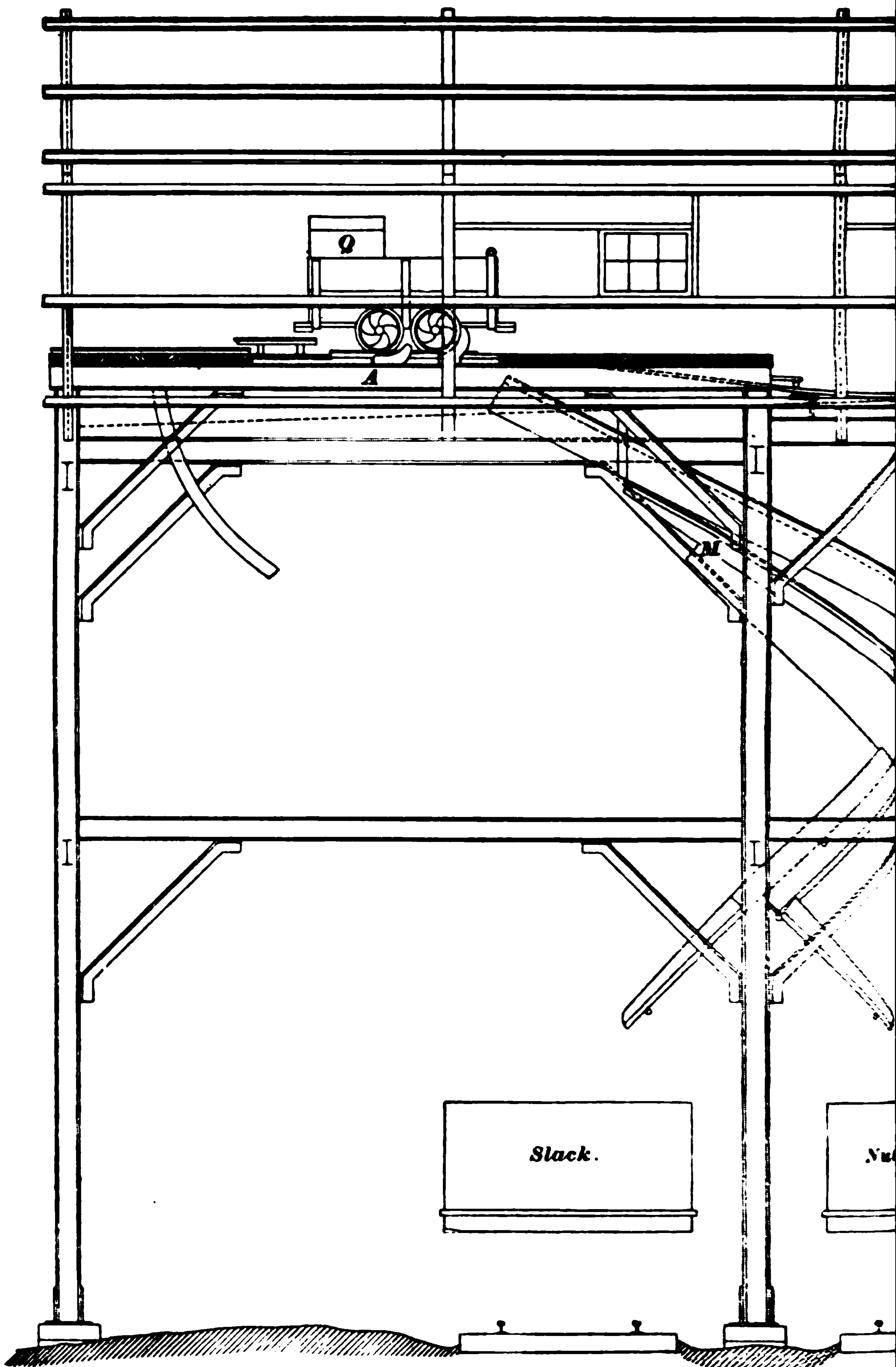
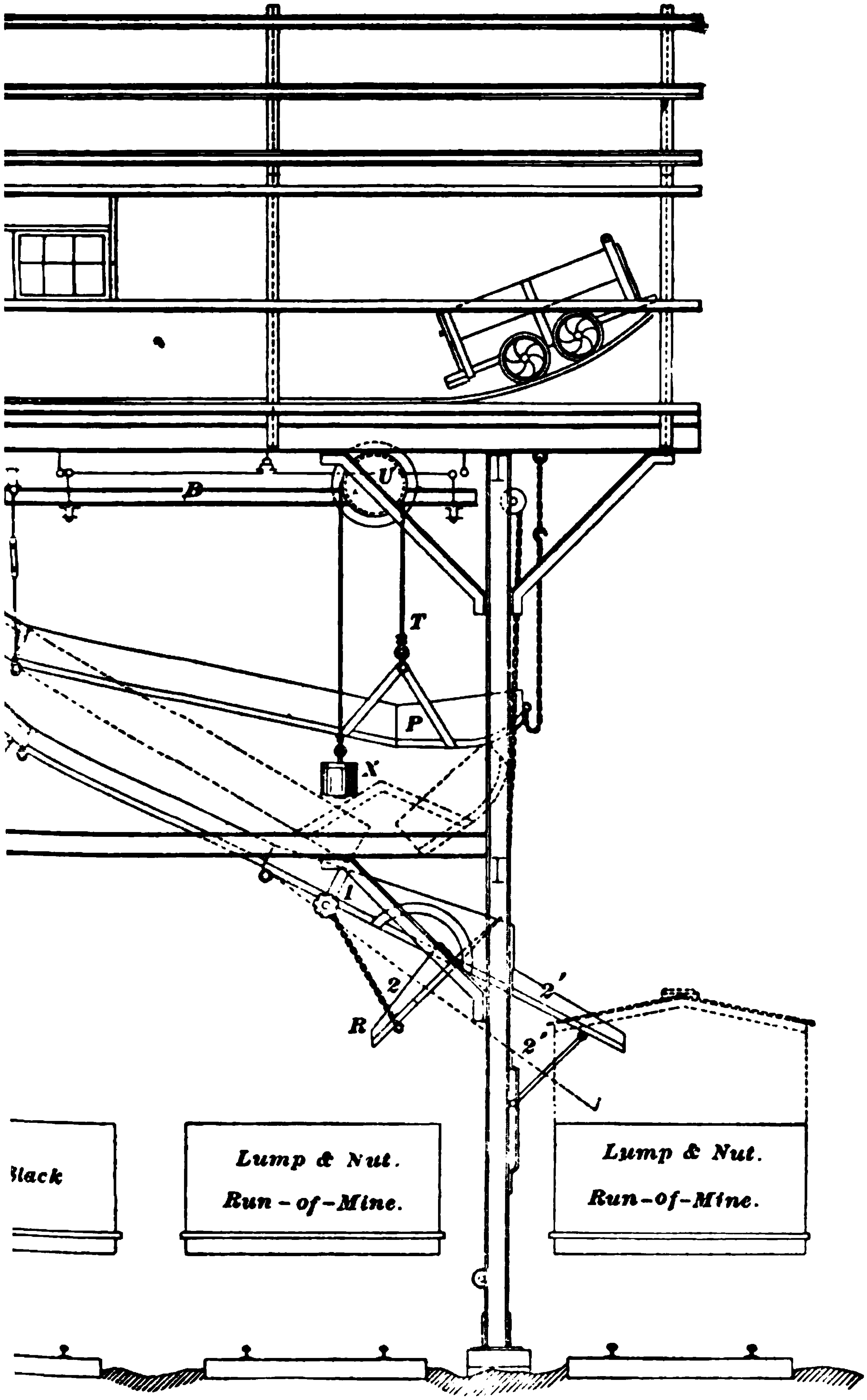


FIG.



B
U
T
P
X
1
2
R
2'
2'
Slack
Lump & Nut.
Run-of-Mine.
Lump & Nut.
Run-of-Mine.

door 2', so that the trimmer can throw, alternately, loads into a box or gondola car on the first track, and into a gondola on the second track. This gives ample time to throw the coal back in the box car without interfering with the loading on the second track.

2662. Where it is desired to produce several sizes, and the elevation of the tipple above the railroad-tracks may not be sufficient, where tracks run between the bents (14 to 16 feet apart) supporting the tipple platform, it may be accomplished by spanning two or four of the railroad-tracks with trusses supporting the tipple platform, which will permit of making the center lines of the railroad-tracks 12 to 13 feet apart, thus bringing the railroad-cars within the reach of the discharge points of two or more screen chutes with a slight elevation of the tipple.

In Fig. 978 the dump is shown at *A*, the screens at *M*, the brake bearings at *B*, the loading basket at *P*, the rope or chain by which it is raised or lowered is shown at *T*, the drum over which the rope or chain passes is shown at *U*, and the counterweight at *X*. The deflecting chutes are shown at *R*. The loaded cars from the mine and the empty cars to the mine are handled in a manner similar to that shown in Fig. 974.

BOX-CAR LOADER.

2663. A **box-car loader** is the name of a device, operated by power, for throwing the coal towards each end of the railroad-car, after it has been chuted into the car door. The loader consists of a pointed ram, oscillating horizontally, which can be moved in and out of the car door. It is located on the opposite side of the lump or mine-run track from the tipple.

When coal is to be loaded, the nose of the ram is moved into the car, and as the load of coal is chuted down against it a double plow-like surface on the nose of the ram oscillates rapidly and throws it to the end of the car. When loaded, the ram is withdrawn. Coal is considerably broken where this machine is used.

WEIGHING.

2664. In some cases the weight of coal loaded in mine-cars is estimated by volume; that is, cars are constructed of such a number of cubic feet capacity as will hold 1½ or 2 tons. This capacity will vary with coals of different specific gravities, and must be determined in each case, especially at isolated mines. This method applies very well for friable coal that will leave no large vacant spaces in the car, and where the coal is loaded level or a few inches above the side-boards. If the coal consists mostly of large sizes, or if the cars are loaded much above the sideboards, the coal should be weighed.

Where the price paid for mining per ton is for mine-run, coal may be weighed in the mine-cars at the tipple platform by passing the loaded cars onto one or two scales, as the arrangement of one or two loaded tracks require.

The scales should be located at a point where the cars can conveniently come to a standstill, and then be readily moved to the tipple. Different locations are shown in the various plans of the surface arrangements, depending somewhat upon the system of tracks.

The scales should be located handy to the tipple, so that the result of the inspection of the coal may be known immediately after its weighing and the miner's check collected.

Where there is no screening, and the mine-run only is loaded on railroad-cars, the weighing can be conveniently done by track-scales, under the railroad-cars, as they stand opposite the chute to be loaded. A possible error in estimating the weight of an empty mine-car is thus avoided. In this case the checks removed from the mine-cars above must be delivered by some device to the weighman below; and if coal from above is run to the boilers or other points, some provision must be made to weigh it, in case weighing is necessary. The arrangement for weighing on railroad-cars can also be applied where only the weighing of lump coal is required, or a combination of the above and of weighing on scales at the tipple platform can be used, where the

weight of lump coal and screenings less than 1¼ inches are each necessary.

It is preferable in this arrangement that the scales below be connected, by a system of levers, to a weigh-beam on the tipple platform, handy to the tipple, so that the weighman can more carefully watch the several operations, and, therefore, be less liable to error in recording the weights of the different products contained in each mine-car.

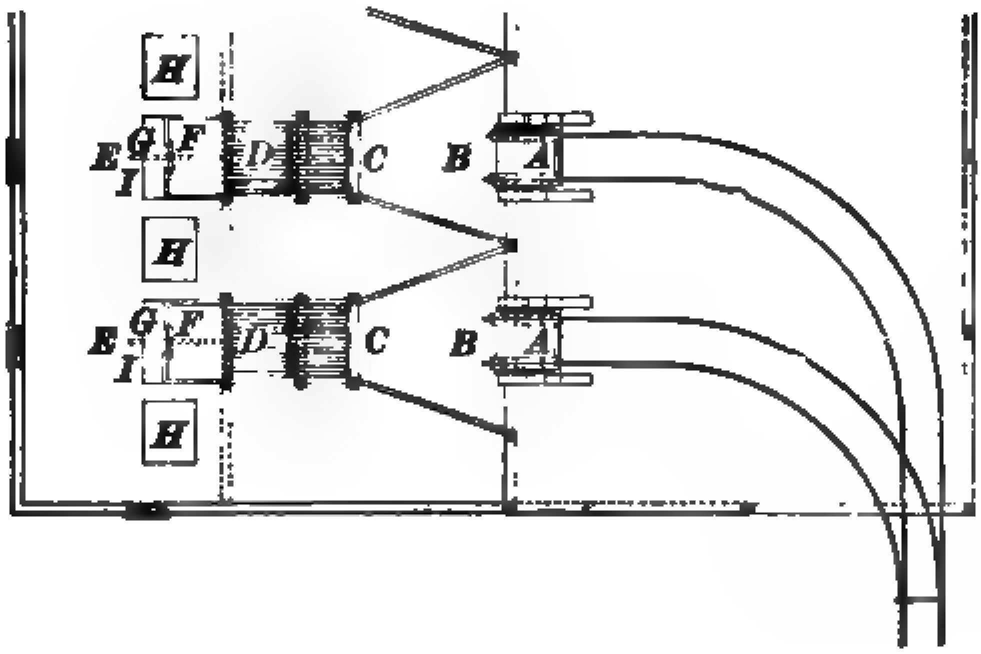
Usually, only the lump coal passing over the screens with 1¼-inch openings is weighed. In this case, a convenient arrangement is shown at *Z*, Fig. 974, and at *B*, Fig. 978, wherein the loading or discharge baskets are suspended to scale bearings, either on or just under the tipple platform, and by a system of levers the weight is transmitted to the weigh-beam *Q* near the tipple horns.

It is sometimes necessary to weigh a certain size of screening that has been separated from the mine-run after dumping, which is to be used for coal washing or coking, and is removed from the bin under the tipple to the latter plants by elevators or conveyors. In this case, the screenings are caught in a hopper resting on scales and connected with the weigh-beam in the platform above.

CLEANING COAL.

2665. If the amount of cleaning that the coal requires is moderate and confined to picking the slate from the lump coal, it will suffice to allow room to lay a narrow track alongside of the lump track, so that mine-cars or smaller ones can be placed alongside of the railroad-car being loaded with lump coal. The slate picked from the coal can be dropped into these small cars, which can be run over a track to some dumping point. If there is considerable slate, rock, and impurities in the lump coal, it will be necessary to plan the tipple especially for the purpose of picking or breaking the lumps to remove it. Fig. 979 shows in elevation and plan an arrangement for this work.

It is desirable that there be as many tipple dumps as


H
E
G
F
I
D
C
B
A
H
E
G
F
I
D
C
B
A
H

possible, so as to deposit the coal at many points, to permit of room and time for a sufficient number of hands to pick over the coal.

The slate pickers are seated along a platform *I*. They regulate the flow of coal down the chute towards them by a lever *E* operating a gate *C*. A small quantity is allowed to flow onto the picking platform, which, after being picked, is shoveled through the coal-hole *G* into the bin *K*, and the slate is thrown into a slate chute *H*. The fine coal from the screen is conveyed away through chute *J*, and the slate is loaded into a slate car for removal through chute *H*. The coal in the bins *K* is loaded into railroad-cars by opening the gate *N*, which is operated by the rope *M* attached to the lever *L*.

Where the nature of the location and existing arrangements will not permit of the width of tipple platform required in this case, and only one or two tipples are used, an endless belt or conveyor, with a sheet-steel surface made in sections, can be introduced at the foot of the chute, and caused to move either in the direction of the tipple or at the right of it, on which the coal from the tipple is received and carried along in front of slate pickers on either side of it. The belt moves slowly, and its length and speed will depend upon the time required to clean the coal. The belt may discharge the picked lump coal directly into a railroad-car or into a bin.

If the nut coal is to be picked, to free it from slate, etc., it should first be passed over a revolving or shaking screen, to obtain a uniform size free from smaller coal. It can then be run in one or more spouts, passing in front of as many slate pickers as may be necessary to clean it. If the nut coal, after screening, is delivered at too low an elevation, it is raised by elevators, located between the railroad-tracks, to a proper height for spouting, picking, and loading into bins or railroad-cars.

If the impurities in nut coal or smaller sizes are considerable, the cleaning can be done thoroughly only by coal washing, which requires that there be not too great a

difference in the sizes of the coal. To obtain close sizes, either revolving or shaking screens should be used, with circular perforations.

HANDLING OF ROCK AND WASTE.

2666. In mines with seams of fair height, say 4 feet upwards, and of clean coal, the amount of rock and waste produced underground that must be hoisted and dumped will not be great. In small seams, or seams with thick slate partings, generally the rock and waste can be disposed of in the gob by some system of mining, as longwall, or in pack-walls in the width of room workings. But even in these cases it may be frequently necessary to hoist rock at times in considerable quantities, should heavy falls occur, or in case of heaving of fireclay bottoms, crushes, and creeps. Provisions for handling rock in these events will greatly reduce interferences with the work.

Storage of rock underground, in some places, is a source of danger from fire, and in such cases it should be hoisted.

A rock dump may be provided outside, where the waste will not be an obstruction, and where a good height for a dump can be secured. This may be somewhere between the mine outlet and the tipple, provided the return of the empty cars to their proper track can be conveniently arranged. If not, a dump beyond the tipple and railroad-track will be preferable, the cars of rock being carried thereto on a trestle over the railroad-tracks. If no available space exists for a rock dump, arrangements may be made for carrying away the rock on railroad-cars, in which case a rock dump can be provided in the tipple, the rock being dumped into a bin, from which it can be loaded into a railroad-car and carried away as desired.

A variety of arrangements for switching out cars of rock as they arrive from the mine is shown for different conditions and requirements in the plans of the surface arrangements of mines. At drift and gravity-plane mines, the rock can be generally switched out immediately after coming out of the mine and before passing to the tipple, as there is

plenty of height for a rock dump on the hillside in front of the plant location.

Cars of rock can be dumped on ordinary tipple horns, which are extended on a trestle as the dumping ground fills up, immediately below the tipple. Rock cars are sometimes run under a derrick provided with a windlass and chain, or rope, with a hook at the end of it, which is put into the eye of the draw-bar at the rear of the car. The rear of the car is then raised by the chain and windlass high enough to let the rock run out. This arrangement does not require tipple horns, the track and the derrick being extended as the dump fills up.

Another arrangement is to erect a set of tipple horns on a truck running on a track about 2 feet lower than the rock track. The car of rock is run onto the tipple horns of the truck, and then with its load hauled to the rock dump, where the car of rock is dumped by the tipple horns tilting on the truck. The truck with the empty car is returned to the starting point and the empty rock car removed.

In some locations the amount of rock or the lack of dumping ground necessitates hoisting the cars of rock up an inclined plane and dumping at considerable height above the level of the surrounding ground, and at some distance from the operations. These cars can be arranged to dump at the side by gates opening in these directions, and the car divided by chutes sloping from the center towards the sides. The rock can be loaded into these cars under bins in the tipple structure, into which the rock has been dumped from cars from the mine or by self-dumping cages.

ARRANGEMENTS FOR HANDLING COAL EASILY BROKEN IN DUMPING.

2667. Fig. 973 shows a cradle for dumping coal that breaks easily from the force with which it strikes the chute. The weight of the load in the cradle causes it to tip, and in so doing it raises a counterweight. When the cradle and car have turned sufficiently to dump, a brake is applied, holding them in this position. The door on the top of the

cradle which holds the coal in the car is opened automatically and releases the coal, which flows out gently into the chute. The door then closes, and, the brake being released, the counterweight causes the cradle to return to its first position, and the car is then removed.

2668. An arrangement is shown in Fig. 980 for lowering coal that breaks up badly from grinding in the chute and from the force with which it strikes the car. This consists of an inclined car *A*, without a bottom, in the chute, its sides resting on the chute and screen bars. The coal

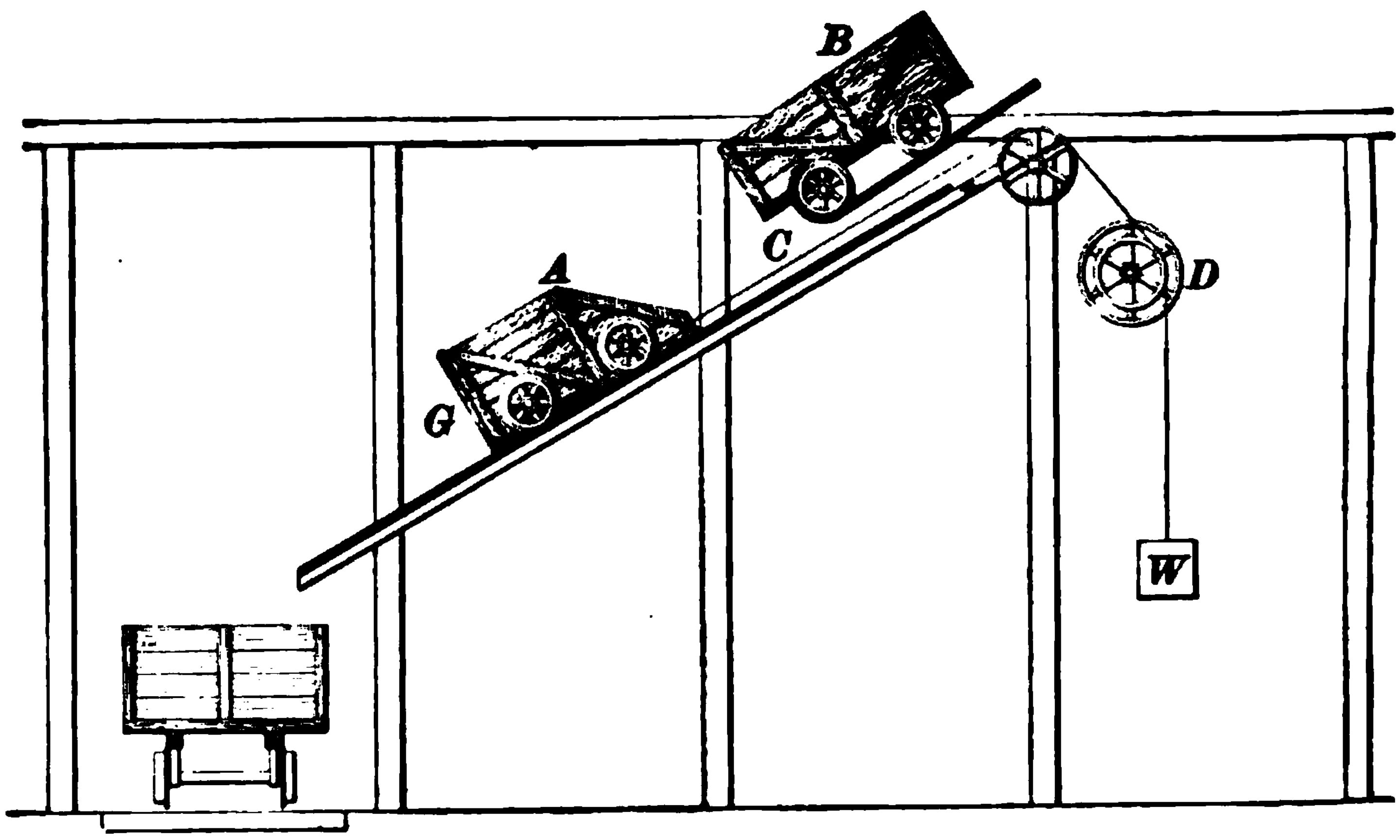


FIG. 980.

from the mine-car *B* is dumped into this, and the weight of the coal in the inclined car causes it to move down the plane, screening the load as it descends. A wire rope *C* is attached to the car at one end, and to a brake drum *D* at the other, the descending car raising a counterweight *W* on the opposite side of the drum. When it reaches the bottom of the chute a brake is applied to the drum, holding the car at the bottom of the chute, while its end gate *G* opens automatically and the load drops into the railroad-cars. The brake is then released, and the counterweight causes the empty car *A* to return to the head of the chute to receive another load.

2669. If there is much fine coal, so that the screening is imperfect in securing large lumps over the ordinary screen bar, an arrangement as shown in Fig. 981 may be introduced. The coal is dumped from a mine-car *A*, either on an ordinary tipple or in a cradle, onto a shaking screen

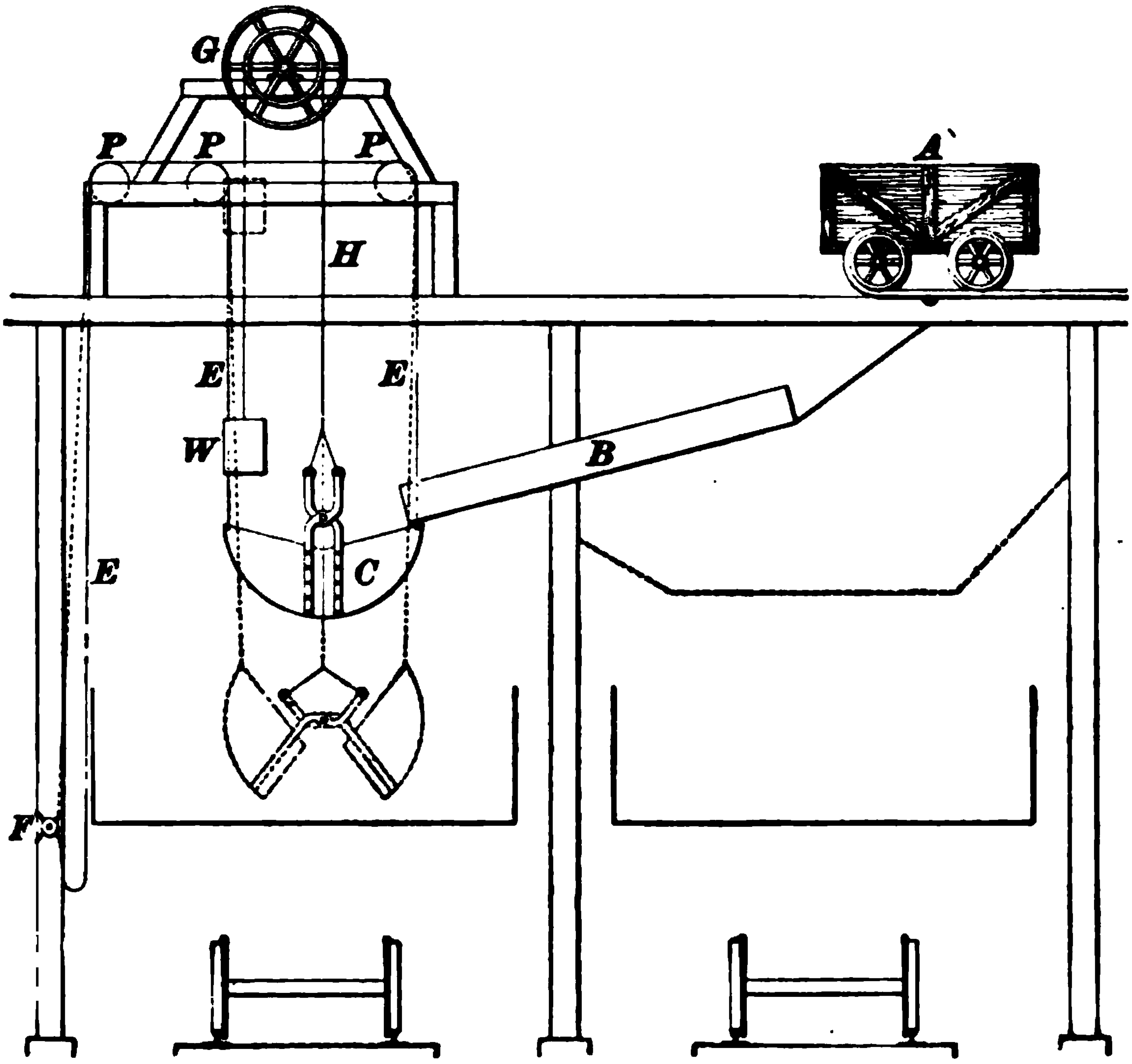


FIG. 981.

B with circular perforations. The coal moves gently over this and is thoroughly screened, and drops into a loading basket *C*, operated similar to that in Fig. 974.

This basket can be arranged to have any amount of play, vertically, between the end of the screen and the railroad-cars into which it is to deposit the coal. The basket *C* is suspended by the rope *H* passing over the brake drum *G* and connected with the counterweight *W*. As the weight of the coal lowers the basket *C*, it is opened by the rope *E E* passing over pulleys *P*, *P*, *P* to the windlass *F*. This rope *E*

can readily be adjusted for opening the basket at any height, so that the coal will not be broken by striking the car. As the load in the car increases in height, the basket can be adjusted to open higher up.

If the height of the mine landing above the railroad-tracks is too great to immediately introduce tipples between the two points, similar to those described, or if a gravity plane can not be used, then an arrangement for vertically lowering the coal may be used, as shown in Fig. 970.

From the lower landing any of the tipple arrangements already described may be used to suit the conditions and requirements.

2670. Arrangement of Tipples for Increasing Dumping Capacity.—Fig. 982 shows an arrangement of

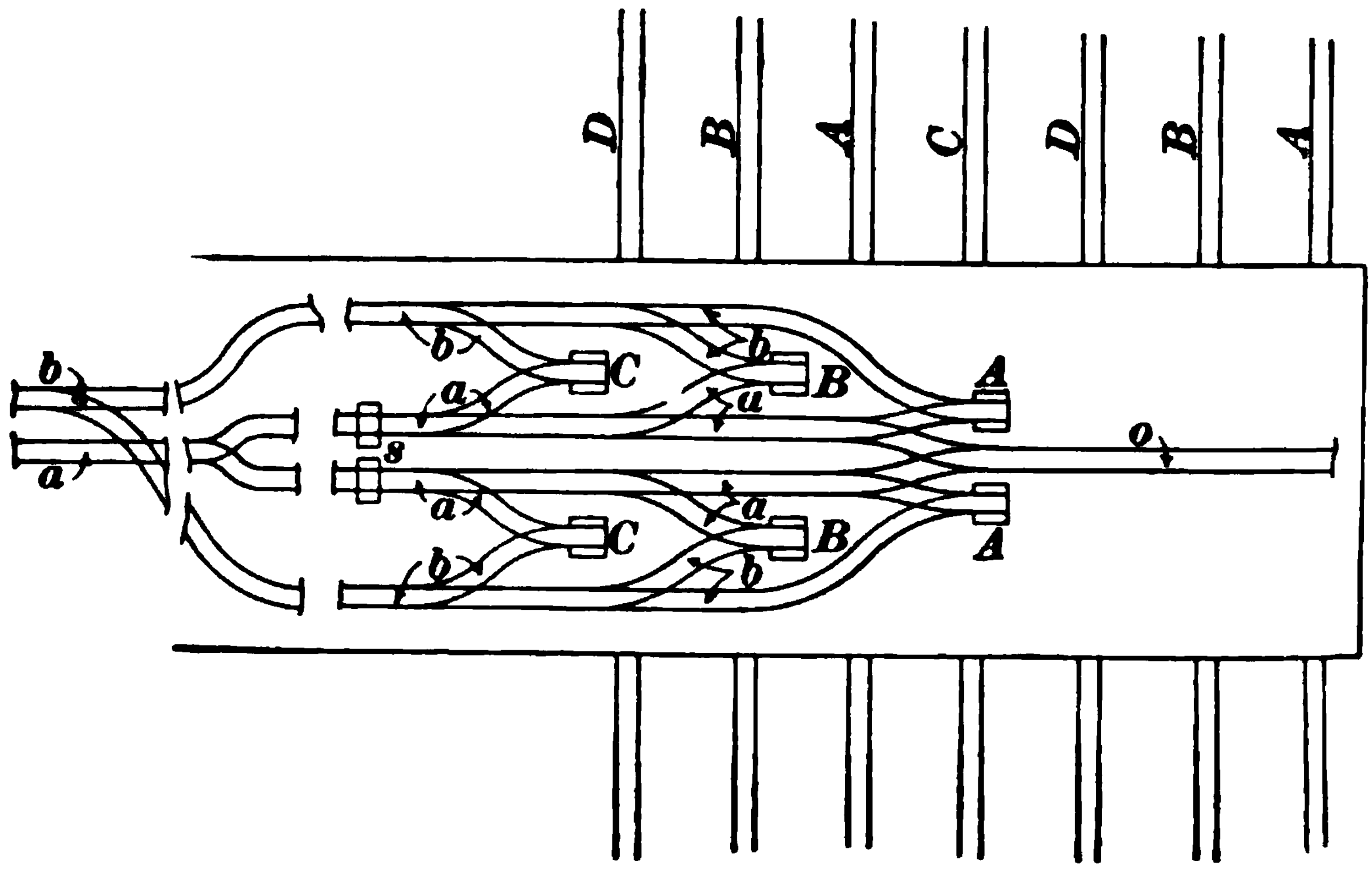


FIG. 982.

tipple with several dumping points for increasing the capacity of the mine and for shipping several sizes of coal, or all mine-run, as desired. On this figure the loaded tracks from the mine to dumps are marked *a* and the empty tracks *b*. The dumps *A*, *A* and *C*, *C* are for coal to be screened, and the dumps *B*, *B* for coal to be shipped as run-of-mine. The scales are shown at *s*. The railroad-tracks are arranged as

follows: *A*, *A*, lump-coal tracks; *B*, *B*, nut-coal tracks; *C*, run-of-mine track, and *D*, *D*, slack tracks.

2671. Construction of Tipple.—Where the trestle is built of timber, posts 10 inches to 14 inches square are used, or the composite posts of 2 pieces 6 × 12 or 7 × 14 form bents for support of the trestle, as shown in Fig. 974. To prevent accidents in case of fire, it is desirable that the tipple be built of steel or iron, especially if near the mine opening, and in any case if it is possible. The roof should always be of corrugated iron.

ARRANGEMENT OF MISCELLANEOUS DETAILS.

RAILROAD-TRACKS.

2672. There should be, in addition to the branch from the main line of the railroad, a track for each size of coal produced, and also a track for receiving material, etc., if the conditions require it.

Railroad-tracks are generally spaced 14 to 16 feet from center to center. They may be less than this; but, to permit of tracks being laid between the bents of the tipple-platform trestle, they should be about this distance, especially if some clearance around the cars is needed for men, or if tracks are on a curve under the trestle.

The curves of standard-gauge railroad-track sidings should not exceed 12° (478 ft. radius), and it is preferable if they do not exceed 7° 30′ curves (764.5 ft. radius). This latter is the usual radius for turnout curves at switches, and requires about a No. 8 or No. 9 frog in the turnouts from straight tracks.

In the case of turnouts with this curve connecting two parallel tracks, 14 feet between centers, the distance from the point of tangency, or beginning of curve to the reversing point between the two tracks, will be about 103 feet, and the distance between the points of tangency of the two tracks will be twice this, or 206 feet.

2673. The length of railroad sidings required at a mine will depend on the output, number of sizes of coal to be produced, and storage room required.

If only sufficient empty railroad-cars are needed to be stored for one day's output, the length of the siding will be found from the following expression:

$$\frac{\text{Daily output of mine}}{\text{Capacity of railroad-car}} \times \text{length of railroad-car.}$$

The storage track for empties may be either single or double track, as the location will permit.

The length of this siding will begin above all the switches leading to the various tracks under the tipple, so that empty cars can be dropped down to any point required for loading, and incoming empties can be placed above the chute without shifting the loads.

The length of the tracks below the chute for each size of coal will depend upon the proportion of each size produced, and the length should be measured in the straight track, not including the switches at the tipple or the switches leading from the tracks into the main empty siding.

If run-of-mine coal is loaded always, or at times, there should be sufficient storage on this track for the whole output, either on a single or a double track. The arrangement of these tracks is shown in the different plans of the surface arrangements of mines.

A repair track, on which to switch crippled empties, is sometimes necessary, and if there is not a material track that can be used for this purpose, a repair track should be provided.

The grades of the empty tracks above the chute should be at least ¾ foot per 100 feet, although 1 foot per 100 feet is better, especially over switches, or if there is much curvature. A grade of one foot per 100 feet, for 200 feet, should be provided at the tipple, to start the cars promptly when loaded. For the balance of the distance the loaded tracks should have at least ½ foot per 100, or ¾ foot per 100 feet is better, especially if there is much curvature. A 1% grade

is sometimes too steep, it being difficult to control loaded cars if they get much headway and brakes work with difficulty.

MINE-CAR TRACKS.

2674. The gauge and distance apart of mine tracks depend upon the size of the cars and other requirements, which vary considerably with the conditions of mining in different places.

For cars with small wheels 12 or 14 inches in diameter, and 18 to 20 inches from center to center of axles, and 3-foot gauge, curves of 12 feet radius can be used; but at least 20 or 25 feet radius should be used, and these should be the sharpest curves for larger cars with wheels 16 to 18 inches in diameter.

2675. It has been indicated what the grades should be for mine-car tracks at shaft and slope landings for short distances, where it is desired to introduce such. For long tracks, where the cars may run by gravity from the mine outlet to the tipple, generally a grade of 16 inches for the first 100 feet, 12 inches for the next 100 feet, and 9 inches per 100 feet for the balance of the distance will be sufficient with track in ordinary condition and where cars with 12 or 14 inch wheels are used.

These grades may be 2 or 3 inches lighter per 100 feet where cars with wheels 16 or 18 inches in diameter are used, and where the track is in good condition and straight, or has very light curves.

The empty cars, returning by gravity, should have a grade of about 1 foot in 100, which should be reduced for the last 100 feet or 200 feet if it is desired to bring them to a standstill before entering the mine. It is important that a steep grade be given at the start, in this case say ½ foot in 20 feet, where the cars start from a standstill.

2676. Where gravity tracks are used, some arrangement, as shown in Fig. 974, is necessary to raise the empties

high enough to return them to the mine, unless they are removed by motor or rope haulage.

It is preferable to introduce this car lift at the tipple, as the tracks near the yard and shops can then be maintained at nearly a level. It is, however, sometimes introduced at the latter point, in which case the tracks at the tipple are nearly level.

Where mule, motor, or rope haulage is used for handling the trips outside for some distance, and it is not desired to maintain a difference in the elevation of parallel loaded and empty tracks, a down grade of about 1 foot per 100 feet will facilitate the movement of the loads towards the tipple, and the empties can be readily returned up the same grade.

The length of mine-car sidings will depend upon the various conditions, as the length of cars, number of cars in a trip, and the frequency of arrival of trips outside, depending upon a long or short haul.

At a shaft mine with landing on a trestle the tracks are planned only with sufficient length for handling single cars, and permit of introducing the necessary turnouts, switches for rock, etc., and return empty tracks. With these arrangements, there is frequently sufficient room for holding several loaded and empty cars, as a reserve in case of slight delays.

If, however, there be some dependent operation, as coke making, for which the coal is dumped into bins to supply the ovens, and they are full, it may be desired to hold the coal. For this, storage tracks for loads and empties sufficient to bridge over usual delays are best arranged underground. On the yard level outside there should be sufficient side-tracks to hold all the mine-cars, in case it is necessary at any time to remove them from the mine.

At a shaft mine with landing on natural ground and some distance between the shaft and tipple, any storage room needed, as above, for loads and empties can be arranged outside. At a slope mine, if provision is necessary for the accumulation of any loads or empties, it must be arranged underground.

The landing room for trips hoisted from the mine should be a little longer than the number of cars in the trip require, and the empty return tracks should have length enough to have one complete empty trip in readiness for return to the mine, and a second one nearly made up.

At a drift mine with a short haul, or frequent arrivals and departures of short trips, the sidings can be short. This is the case where the arrangement of the haulage tracks inside permits of moving several trips in either direction at the same time, as with mules, endless-rope, or locomotive haulage. Long sidings, however, are necessary where long trips are moved, on account of long haul, and either the above systems or tail-rope haulage is used.

2677. The method of determining the length of siding required can best be ascertained by taking the following examples:

Suppose the cars to be of 1 ton capacity, the trips to arrive every 20 minutes, and the mine to work 8 hours per day, then $\frac{8 \text{ (hr.)} \times 60 \text{ (min.)}}{20 \text{ (min.)}} = 24$, the number of trips per day.

If the output is 1,500 tons a day, then $\frac{1{,}500 \text{ (tons)}}{24 \text{ (trips)}} = 62 =$ the number of tons hauled per trip.

If the cars are of 1 ton capacity, then there must be 62 cars on the trip.

If the cars are 7 feet long, then the length of the siding for the arrival of the trip should be $62 \times 7 = 434$ feet long.

In the calculation of the length of mine-car tracks the uncertain factor of delays and accidents makes it desirable to allow double the calculated length or more, if the expense can be incurred and the location will permit, so that double the above length, or (434×2) 868 feet, of siding should be allowed for both the loaded and empty track.

2678. To find the number of cars required for mining operations, the following example may be taken as an illustration:

Estimating the length of haul underground as 1 mile, at

a speed of 6 miles per hour it will require 10 minutes to make a trip in one direction; the total time for a round trip will be as follows:

Hauling empty 1 mile......................	10 min.
Distributing, loading, and gathering cars....	1 hr. 00 min.
Hauling loads 1 mile	10 min.
Dumping and returning to mine	20 min.
Contingencies	20 min.
Total................................	2 hr. 00 min.

Therefore, if it requires 2 hours for a car to make a round trip, a car will make only 4 trips a day

If the capacity of the car is 1 ton, then 1 car will handle 4 tons a day, and it will require (1,500 tons daily output ÷ 4 tons) 375 cars to handle the output.

2679. In laying out a turnout, or in connecting one straight track with another, the following rule will determine the distance between the switch and the frog: The distance from the switch, or point of curve, to the frog may be found by multiplying twice the radius of the curve by the gauge of the track and extracting the square root of the product. Or, by referring to Fig. 983 for the meaning of the letters, it will be seen that the rule can be briefly expressed in the formula

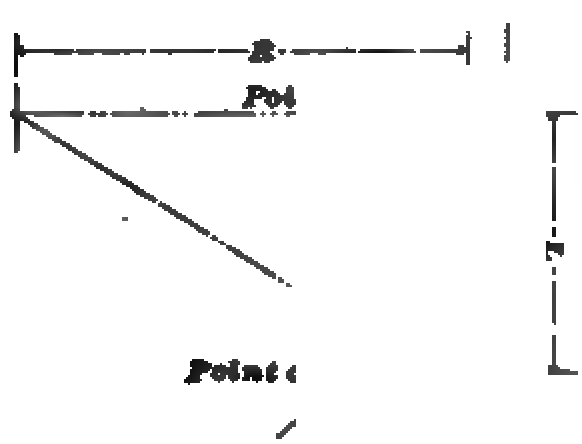


FIG. 983.

$$L = \sqrt{2 R d}. \qquad \textbf{(214.)}$$

COAL-WASHING PLANT.

2680. If a washing plant is necessary for cleaning coal for shipment or for coke-ovens, it may be located near the tipple, and the sizes produced in screening delivered there by elevators and conveyors removing the coal from under the screens of the coal tipple.

If the coal washer is located at some distance from the tipple, it should be along the railroad-track. The coal is best conveyed there as one size, generally as coal less than 1½ inches in size, either by conveyors or railroad-cars, which deposit it in a pit, from which it is lifted by an elevator, and then separated at the washer plant into the sizes best suited for washing. The coal-washing plant should be handy also to the head of the lines of coke-ovens if the cleaned product is to be used for coking.

COKE-OVENS.

2681. Beehive coke-ovens are generally 12 feet in diameter and 6½ or 7 feet high. They may be built either in single rows or in blocks. The latter are shown in Fig. 984. They should be located so that the charging larry *B*, after being loaded at the coal-bins *A*, can be run down the track on top of the ovens, which has a grade of at least 1 in 100. This track should be supported on piers and to one side of the tunnel head of the ovens *C*, so that the larry will not be exposed to the heat of the ovens. By referring to Fig. 984, the coke wharf *D*, with a height of 8 feet, will be seen, with coke cars *E* in position for loading. The larries are moved over the tracks by an endless rope *G* operated by an engine located under the tracks at *F*.

It is preferable that the coke-ovens *C* be situated on the same side of the railroad-track that the mine is, but some distance away. If they are built beyond the railroad-tracks, a trestle can be built from the landing platform of the mine, over the railroad-tracks, to the coal-bins of the coke-ovens.

If the location for the coke-ovens does not permit of a down grade for the loaded larry, or if an opposite grade is necessary, rope haulage can be introduced, as shown in Fig. 984. Loaded larries may be hauled up a grade of 3% or more with an arrangement of endless-rope haulage, as here shown.

A plant of 100 ovens, 12 feet diameter, in a single row,

requires a width of 48 feet and a length of about 1,600 feet, with bins, etc.

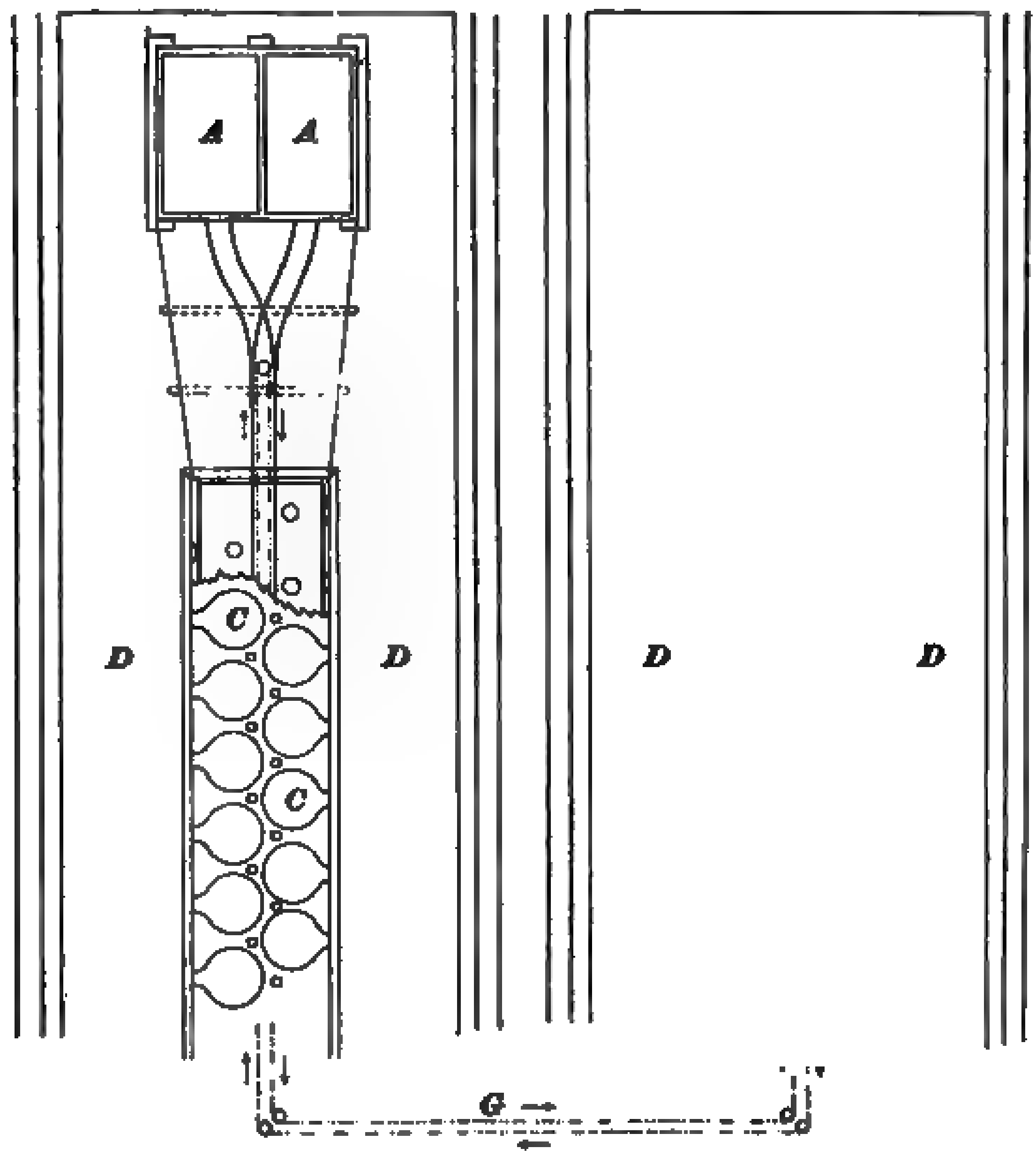


FIG. 98A.

A plant of 100 block ovens requires a width of 98 feet from the center of the railroad-track and a length of about 900 feet, with bins, etc.

The charge for a coke-oven is about 5 tons. One hundred ovens running on 48-hour coke will require 250 tons daily. The capacity of the bins should be about equal to this.

Estimating 42 cubic feet of coal to a ton, the contents of the bins must be 10,500 cubic feet.

A height of 50 to 60 feet is needed from the railroad-track to the top of the coal-bins at the coke-oven plant. The tipple landing should be arranged on a level with the top of the bins, if possible, where mine-run is coked.

If, however, screenings are coked, the above arrangement is not of such importance, as the screenings will generally have to be raised by elevators to the bins.

If the tipple platform can not be arranged on a level with the top of the coke-oven bins, or if the elevation of the ground for the site of the coke-oven plant is not sufficiently low to make the top of the bins on a level with the tipple platform and lead the railroad-tracks to the ovens, then it will be necessary to arrange a short incline for hoisting the mine-cars from the tipple platform to the top of coke-oven bins.

WATER-SUPPLY.

2682. The water-supply may be obtained as follows:

1. From the mine water, if in sufficient quantity to be pumped from the mine to a reservoir, provided the presence of sulphur, alkali, and impurities does not exclude its use. If the water is dirty or contains suspended matter, it will require settling before use.

2. From springs or streams at higher elevation than the mining operations, the water from which may be held in a reservoir at sufficient height above operations to afford sufficient head for pressure at the works.

3. From sources lower than mining operations or from deep wells, requiring pumping stations to force the water through a water-main to a reservoir at sufficient height above the operations for pressure.

In the erection of a pumping station, the site should be considered with reference to its location near tracks for the

supply of fuel, or, if at a distance, with reference to the means of getting coal to the plant.

If mine water can be used in any of the operations, as for boilers, coal washing, or coking, it should be held in a separate reservoir from the fresh-water supply. Water with some sulphur may be unfit for boilers, but suitable for coal washing or coking, although sometimes there may be too much sulphur in the water to permit of its being used for any purpose.

2683. Whatever may be the sources of the supply, the water will be stored or collected in reservoirs, made as follows:

1. Made by an earth dam or breastwork thrown across a ravine. The inside may have to be puddled with clay or cemented, to make it hold water.
2. In tanks of wood, iron, or stone.

Generally, no more than two reservoirs will be needed, and one will be sufficient if only a fresh-water supply is stored.

2684. Style and Location of Reservoir.—Where a reservoir can be readily constructed by an earth dam at sufficient elevation, and the cost of pumping or collecting water therein is not very expensive, such will serve as well as a wood, iron, or stone tank. If, however, the cost of pumping is expensive, owing to the distance of a pumping plant from the reservoir, scarcity of water, and its evaporation or absorption in an earth reservoir is considerable, then a wood, iron, or stone tank should be used. Wooden tanks are erected for temporary and iron or stone tanks for permanent service.

Where there is no ground in the vicinity of the operations at a higher elevation than the plant, either a wooden or iron tank should be erected on a trestle sufficiently high to give the desired pressure at the works. The pressure need not be sufficient to feed the boilers, as a feed-pump will generally be used for this work.

Where there is ground at sufficient height for the location

of a reservoir, either a wood or iron tank can be used, and need not be located on a trestle. Or, if stone is plentiful and the nature of the soil will permit, a stone reservoir can be constructed, affording permanent service.

If there is much sulphur in the water, it precludes the use of an iron tank, unless coated inside with paint or some substance to resist the action of the sulphur.

2685. Construction of Earth Dam and Reservoir.—In the excavation for a basin or the building of a dam, it is preferable that the soil be clay and well tamped as its building progresses.

If the soil is sandy or gravelly, the building of a dam may be impracticable without borrowing suitable material elsewhere at too great an expense.

If the soil can be used, it may be necessary after completion to give the surface of the reservoir a heavy layer of concrete, with a coating of cement.

The inner slope should be 2 to 1 and the outer slope $1\frac{1}{2}$ to 1. The top of the embankment should be 8 to 10 feet wide, although this varies with the height of the reservoir, and may be found from the following expression:

$$T = 2 + 2\sqrt{H}, \qquad \textbf{(215.)}$$

where T = top width;
H = height of water.

In small reservoirs the water should be about $2\frac{1}{2}$ or 3 feet lower than the top.

2686. Wood and Iron Tanks.—These are procured ready made, ready for erection. The foundations of wood tanks generally consist of 4 parallel bents, braced together, and with stringers laid across the bents, on which are placed $2'' \times 12''$ floor joists supporting the bottom of the tank.

The foundations of iron tanks are generally of stone, built with 4 parallel piers, across which are laid steel rails to support the bottom of the tank.

2687. Construction of Stone Reservoirs.—For heights of 8 to 10 feet, the inside of the wall may be straight

and the outside battered 3 or 5 inches to the foot, depending upon the weight of the stone used. The top width will be about 18 inches. The bottom of the reservoir should be built with stones laid closely together and well filled with mortar or concrete. The side walls are then built up on the bottom foundation, and, when complete, the whole interior is coated with ½ to 1 inch of cement.

2688. Amount of Water Needed.—This will depend upon the quantity of water needed in each of the different operations. In determining the amount of water-supply for a coal plant, the following needs must be considered:

1. Supply to boilers; 30 to 35 lb. of water per H. P. per hour should be allowed.
2. Household use.
3. Shops and stables outside.
4. Underground stables and drinking.
5. For air-compressor; cooling air.
6. For coke-ovens; allow 300 gallons per oven watered daily.
7. For coal washing. The requirements per minute will depend upon the size of the plant. A fair-sized plant uses 400 to 600 gallons per minute, but, as most of this is water that is used over and over, provision need be made only for supplying 50 to 100 gallons for replenishing loss per minute.
8. For fire purposes outside and underground.

Mine water may possibly be used for these three latter purposes.

2689. Arrangement of Pipes.—The reservoir is filled with water by a feed-pipe from the source of supply flowing into the reservoir over its top. An outlet or supply pipe supplies the works and other points of consumption, and draws water from the bottom of the tank.

This pipe is laid from 3 to 6 inches above the bottom of the reservoir.

A pipe is also laid level with the bottom of the reservoir,

with a valve on the outside for cleaning the reservoir when filled with dirt up to the supply pipe.

Sometimes the feed-pipe filling the reservoir is also made a supply pipe to the works, tapping the reservoir at the bottom. In this case, if the water is pumped to the reservoir from a pumping station, a valve outside the reservoir can be turned off, checking the inflow of water to the reservoir and permitting the full pressure of the pump to act on the supply pipes.

If there are separate feed and supply pipes, a connection with a system of valves can be made outside, so that, by turning off a valve on each of the feed and supply pipes and opening a valve in a short pipe connecting the other pipes, a direct pressure from a pumping station supplying the reservoir can be obtained in case of fire.

If the above arrangement can not be made, the supply pipe to the important points to be guarded against fire should be laid, so that connection can be made with a pump in the boiler room to give the necessary pressure.

2690. Size of Pipes for Main Reservoir.—The size of the feed and supply pipes will depend upon the extent and nature of the operations. An ordinary-sized water-main is 3 or 4 in. in diameter for a mine and the requirements connected therewith, using six boilers, 70 to 100 H. P. each, requiring a reservoir capacity of about 30,000 gallons, or possibly 50,000 gallons capacity in case of accident to the pumps.

If the operations include 100 coke-ovens, a 4-inch water-main should be laid, allowing about 15,000 gallons additional tank capacity.

Where extensive coking operations and coal washing are carried on, a 6-inch water-main and a 150,000-gallon reservoir will not be too large. Pipes should be laid below the frost line and well covered with soil. Around the mine buildings, coal and rock dumps, and coke works, they should be well surrounded with clay, so that sulphur in the coal or water can not destroy them.

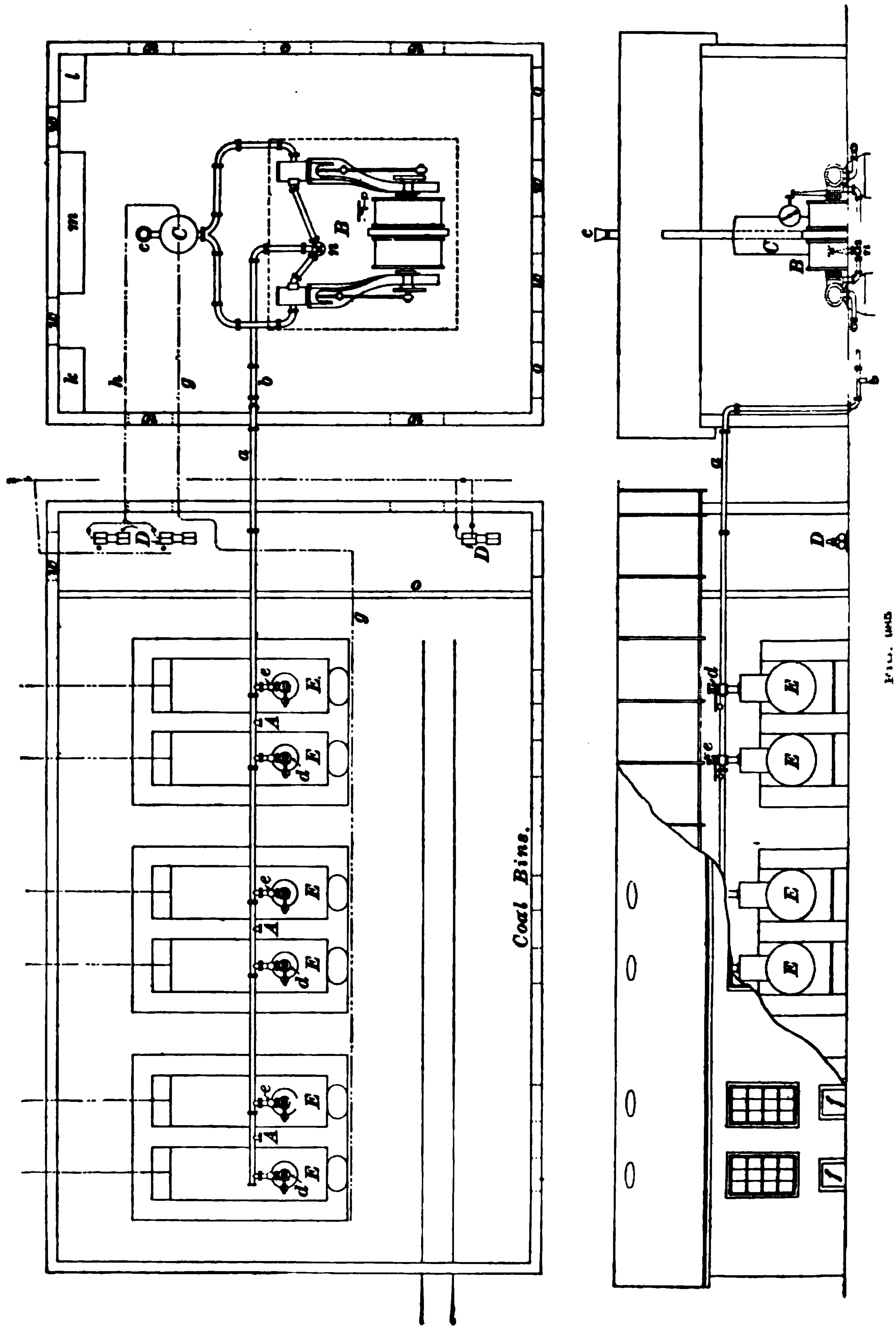


FIG. [illegible]

ARRANGEMENT OF BUILDINGS, SHOPS, ETC.

2691. Engine Room.—A building about 34 × 46 feet inside will be large enough for an engine. This will allow plenty of room between its walls and the engine to conveniently work about the latter. It should be well lighted, and the windows arranged so that the engineer can command as full a view of the landings, etc., as possible.

Fig. 985 shows an arrangement of an engine room with feed-water heater *C*. The steam to the engine *B* is admitted by a throttle-valve *n* below the floor. This prevents heating up of the engine room and delivers drier steam to the engine than where the throttle is over the head of the engineer.

If much water is carried over with the steam from the boilers *E*, a steam separator should be introduced in the steam main just before reaching the engine.

Generally, if the main *a* is large enough, any condensed water can be caught in a drip-pipe *b* and blown off occasionally.

If the location will permit, it is a good plan to locate the boilers low enough so that the steam main will drain back to the boilers from the throttle of the engine.

2692. Boiler House.—A convenient arrangement for a boiler plant is also shown in Fig. 985. The depth of the house is 46 feet. Its length will depend upon the number of boilers.

For six boilers its length should be about 64 feet, with 10 feet at one end for a feed-pump room.

Sufficient space should be allowed in front of the boilers for stoking, removing ashes, handling of rakes, flue cleaners, and removing flues, etc., and at the rear of the boilers for cleaning, repairs, etc.

Boilers may be built solidly together in a battery, although it is best to separate them in nests of two, as they can be better cooled off for examination and repairs.

The stacks for boilers are generally No. 12 or 14 sheet iron, 28 to 32 inches in diameter, one for each boiler.

Sometimes a brick stack is erected, with flues leading to it from the boilers. In this case space should be provided for the foundation of the chimney, which for six or eight 80-horsepower boilers will be about 13 feet square at the base and 11 feet square at the top, with a flue 5 feet square lined with firebrick.

Two feed-pumps *D* should be provided, and possibly a fire-pump *D'*. The feed-pumps should be arranged to act together on the same supply and feed pipes to the boilers, or independently, if either is disconnected. At each boiler there should be a short branch from the feed-pipe, provided with a globe and check valve.

The arrangement of a safety-valve *d* and gate-valve *e* for each boiler, and their connection with the steam main *a*, is shown in Fig. 985. The latter also has branches for connection with other machines to be run by steam at *A*.

The location of the boiler house should be central to the principal points to which it is to supply steam. Generally, a location is best by the side of the hoisting-engine, or it may be at the rear, if this location is not too far removed from other engines.

One side of the boiler house should be free, for the removal of boilers for repairs. It is preferable not to sink the boiler house lower than the level of the surrounding ground, but to keep it somewhat elevated or where there is a fall in the ground, for the disposal of ashes.

Boilers should be located so that it will be convenient to obtain their coal supply from the mine-cars; or they may be located near the railroad, preferably the slack track, and draw their supply of coal from there.

Coal-bins should be located in front of the boilers, with coal-holes on a level with the floor of the boiler house.

If the location of the boilers does not permit of running the ashes out on a tip car to an ash dump, it will be necessary to arrange a pulley hoist for raising ashes in a bucket to the tipple platform, where they are dumped into cars to be run to the rock dump.

Ashes may sometimes be needed for ballasting roads in

the mine, in which case the ash track is connected with some track leading into the mine. By referring to the figure, it will be seen that the feed-water heater receives the exhaust-steam before it is discharged from the exhaust-pipe *c*. The coal-holes in the boiler house are shown at *f*. The water-main is shown at *i*, the cold-water pipes to the feed-water heater at *h*, and the feed-water pipes from the heater to boilers at *g*. A closet for oils, etc., is shown in the engine-house at *k*, and one for clothes is shown at *l*. A work-bench, with drawers, etc., is shown in the engine-house at *m*. Windows are shown in the plan at *w* and doors at *o*.

2693. Air-Compressor Building.—The air-compressor building should be about 30 × 40 feet. This will allow the introducing of three compressors, 22″ × 24″, or the same building will be wide enough for two 26″ × 32″ compressors.

The former require a foundation space of 5′ × 21′ 6″. There should be 4 ft. space between the foundations of air-compressors, and about 10 ft. should be allowed from the ends of the compressors to the walls of the building, or more, depending whether there are any long rods to be drawn out at the steam-cylinder end for repairs. At the air-compressing end an arrangement is necessary for an inlet of fresh air drawn from outdoors, if the indoor air is not cool enough.

An air receiver will require some space in a corner of the building, unless it is located underground.

Connection from the water-supply is necessary to furnish water to the water-jacket or pipes of the compressor for cooling the air.

2694. Electric-Power Plant.—A building 30 × 40 feet will be of sufficient size for generating machinery of about 200 to 300 H. P., where power is to be furnished for ordinary haulage, coal cutting, and pumping at distant swamps and dips underground. The size of the building will vary from this to about 60 × 100 feet, where the generating machinery is from 600 to 800 H. P.

In smaller plants there may be one or two driving engines, each connected with two 40 to 80 horsepower generators, which are belted directly to two fly-wheels on an engine, the belting being about 17 to 26 feet from center to center of shafts.

In larger plants, the best arrangement is to have two engines, each of sufficient power to run the whole plant, and each connected with belting to the same countershaft, on which are pulleys that may be belted to six or ten generators of from 40 to 80 H. P. The countershaft may be in two parts, and connected at its center with a clutch pulley, so as to throw out half of the plant when not needed.

The distance from the center of the engine shaft to the countershaft will be about 30 or 40 feet, and from the countershaft to the center of the generator shaft about 17 to 26 feet.

The location of the plant is not necessarily very near the mine opening, and if there is any convenient water-power in the neighborhood, it should be located at that point.

If the cost of the plant can be incurred, and the use of electricity is not excluded by the presence of gas where the wires are located, or other conditions, it may be used for haulage underground.

Electricity is also used for coal cutting and pumping at distant points underground, where the conditions permit of its use.

2695. Blacksmith Shop.—This may vary in size from about 15 feet square, for small operations, to about 30 feet square for more extensive work.

The location should be convenient for receiving tools for sharpening from the miners as they go to and from the mine, or for sending them there from the inside.

It should be alongside of the carpenter shop, for handling irons back and forth for drift-car repairs; also near the machine-shop, for repairs to machinery needing work done in both places.

The convenience for shoeing mules at the stable outside or underground should also be considered.

There should be 1 to 3 forges and anvils. Blast should be furnished to forges by a blower driven by some existing power, if convenient.

The shop should also be provided with proper work-benches, closets, and racks for pieces of iron, drill-press, and an iron bender, if the work requires it.

2696. Tool House.—A tool house 15 × 15 feet or 15 × 30 feet, as required, may be provided in a building adjoining the blacksmith shop or near it, where miners' picks may be placed on racks, before and after sharpening, also the drills and other tools of rock gangs and other day hands.

If the tool house is mainly for holding shovels, picks, etc., of outside laboring gangs who use them for cleaning coal, dumping rock, repairing track, or handling material, its location at a point central to this work is to be desired.

2697. Carpenter Shop.—This will be from 15 × 20 feet to 30 × 50 feet in size, according to amount of work and drift-car repairs to be done. It should be located handy to the mine and near the empty return tracks for receiving crippled and despatching new or repaired cars. It should be convenient to the blacksmith shop for work required on car iron, and near the lumber-yard. It should be provided with necessary work-benches, closets for tools, and bins for nails, bolts, screws, and mine-car fittings.

A stock of mine-car wheels and axles may be kept in a shed near the carpenter shop, unless such are stored in the iron house. A grindstone should be provided, preferably outside the shop, so that miners and employees can use it.

If rollers for rope haulage are to be turned, there should also be a wood-turning lathe, which should be run by power, if convenient.

2698. Machine-Shop.—Ordinarily, a site central to the surface machinery should be selected for the machine-shop. This is most advantageous where operations do not involve much machinery, and the number of machinists and

helpers is few, and their duties can include oiling, attending the fan-engine and other machinery not requiring continual attention.

Where operations involve more machinery, and the work is sufficiently extensive to require continual attendance of machinists and helpers, each to his special line of work, as bench machinists outside and pipe-fitters underground, the location of the shops may be at any convenient point, so that machinery from the mine can be readily conveyed thereto and the outside machinery readily attended to.

There should be plenty of room outside the shops for the depositing of machinery needing repairs, or that which is again in readiness for use. This space is also useful for pipe-fitting and the testing and repair of quantities of pipe that have been removed from the mine and accumulated; also for the uncoiling of ropes for repairs and splicing.

The location should also be near the blacksmith shops, for the convenience of work which requires being handled in both places, and near the stock of iron piping and supply room.

If it should be necessary to have machinery in the shop run by power, its location should be studied with reference to transferring power by line shafting or belting from some existing engine, as one operating a saw, shaking or revolving screens, forge blower, or inclined hoist for lumber and material. The advantage of a location for readily handling the work should have first consideration, even if an independent engine is necessary to furnish the power.

Where the amount of work is small, the shop need be large enough only for a working bench, racks, shelves, and closets, for the convenient arrangement of machinists' tools and such few supplies and fittings required for immediate needs. A building 12′ × 20′ will be large enough in this case.

It will be necessary to increase the size of the shop with the increase of machinery in mining operations and work required for its maintenance and repairs.

A large machine-shop is not necessary, except in isolated

locations, where, due to distance and time required in sending for and receiving parts for repairs, the operations would be seriously delayed or interrupted. In this case, it is rarely necessary to provide for repairs to large parts of machines. Generally, the parts most subject to breakage are kept in duplicate, and it can be arranged with the manufacturers of special machines to use a telegraphic code for the prompt shipment of large duplicate parts, which will generally be delivered in about as short a time as they can be turned out by an extensively equipped machine-shop, furnished and maintained at an expense that in the end will amount to more than may be involved by any delay to operations in the time required to secure large duplicate parts by a telegraphic code.

A shop 30′ × 50′ may be large enough for the requirements of mines utilizing machinery to the fullest degree, and where machine-shops exist in the region doing custom work, to whom special work may be assigned.

The usual machinists' tools should be provided, a set for each man or helper, including such tools as he needs. Benches fitted with vises, closets and shelves for tools and supplies, racks for larger fittings and tools should be of such capacity as the needs of the mines may require. Stock and dies, pipe-cutting and threading tools, and drill-press should be in such sizes as the work requires.

Tools should be at hand for properly handling such work as may arise with boilers, engines, pumps, fans, etc., that may be in use. Where drift-car irons are shaped, a punch for bolt-holes is desirable.

An emery grinder saves file work, where much sharpening of saws or drills is required.

If locomotive, compressed-air, or electric plants are used in the operations, the repairs attendant to these machines, and the advisability of undertaking small repairs to pumps and engine valves and cylinders, may make it necessary to have also a small lathe and planer.

2699. Material and Lumber Yard.—If much lumber and material are received by railroad, it is desirable,

where the conditions will permit, to unload at some convenient point from the railroad-car, and at the same time be near the mine opening, carpenter shop, and saw.

If the most convenient location for a lumber-yard near the mine is at some distance from the railroad-tracks, or at a higher elevation, the means for conveying the material thereto, under different conditions, has been indicated in the various plans for surface arrangements.

The lighter lumber, which is to be cut at the saw or for use in the carpenter shop, should be nearer these points. Props, if they are delivered already sawed, and other heavy lumber should be deposited nearer the mine than the saw. Steel rails will generally be deposited in the material yard, and scrap-iron should be gathered and deposited at some convenient point or in the material yard, to be shipped as required. Space can also be provided here for the stock of brick, lime, and sand needed in the operations.

2700. Sawmill.—A circular saw is generally needed in isolated locations, or where much cutting of lumber of various sizes is required from stock lumber received from a distance, or where standing timber is to be cut.

It should be located handy to the carpenter shop and to the lumber-yard near the mine, unless it is desired to have it convenient to standing timber for cutting.

2701. Supply Room.—The size of the supply room will be from 20 × 20 feet to 30 × 50 feet, or larger, depending upon the nature of operations, machinery, and the variety and quantity of supplies it is necessary to have on hand.

The stock will include, principally, miners', carpenters', and blacksmiths' tools, unless they are carried at a store; also nails, screws, bolts, spikes, mine-car fittings, brattice cloth, harness, and fittings and parts for repairs to such machinery as engines, boilers, fans, pumps, compressed-air or electric machines, belting, steam and water pipe fittings.

The supply room should be located handy to the shops for issuing supplies, and also for the receipt of supplies by railroad or otherwise.

2702. Iron House.—Generally, a building 12 × 24 feet will be of sufficient size for storing pipe, round and bar iron, tool steel, sheet iron, wheels, axles, etc. It should be located handy to the blacksmith and machine shops.

Piping may be stored at the material yard; wheels and axles may be stored there or at the carpenter shop.

2703. Oil House.—Oil of all kinds may be kept at a store, where it is sold to miners, and issued in stated quantities at intervals for mine use, drift cars, machinery, etc.

In this case, there should be small oil-cans having a capacity of from 1 to 10 gallons. These sizes will be suitable for the various requirements.

These should be kept on iron trays to catch all leakage, and securely enclosed, and at a safe distance from waste and other inflammable materials.

If the oil house is located near the mine, it should be handy to the storekeeper for issuing, but at some distance from surrounding buildings. It may thus be issued in smaller quantities for machinery and mine use, but the same precautions must be taken wherever it is used.

Oil may be held in barrels in the storehouse, but it is safer if transferred to iron tanks of about 20 barrels capacity each for miners', black, and coal oil. Special engine and cylinder oils may be stored in tanks of 2 to 5 barrels capacity.

In warm climates it is advisable to transfer oil to iron tanks to prevent loss from leakage in barrels.

If the heating of car oil is necessary in cold weather, it should be done on a coil of steam-pipe, if the point at which cars are oiled is near the tipple or other structure.

2704. Hay and feed should be stored handy to the stables and to storekeeper for issuing. At shaft mines, the feed should be conveniently reached by a track, so that feed can be loaded on a mine-car at the storehouse and hauled to the shaft for lowering to the stables underground. This should also be the case if feed is sent underground at noonday, at mines opened otherwise. The feed should be stored

at some distance from other buildings, as a precaution against fire.

2705. Stable.—The stable should be located so that mules can readily pass between it and the entrance to the mine, and handy to a wagon road leading to points where hauling may be necessary. It should be handy to the blacksmith shop, for shoeing mules. Where it is possible, stables should be located where a yard or pasture can be enclosed for mules needing rest. A harness and wagon room should be located alongside of the stable.

2706. Powder House.—This may vary from 8 feet square to 16 × 20 feet, for holding small quantities of explosives or one or two carloads. It should be built on the opposite side of a hill from the mine, if possible, and not less than 1,000 feet, and a mile away from other buildings is preferable, if the explosives are in large quantities and can be reached by a wagon road. The building should be of brick, stone, or iron, and should have an iron door. There should be an opening in the door, and another at the end of the building, for ventilation and to prevent explosives from absorbing moisture from the ground, etc.

2707. Wash-House.—This may be located at any place where space will permit, convenient for employees, or near the shops, or on the slope of some ground where the drainage will be perfect. It is sometimes located near the boilers, with the view of drawing hot water therefrom, but this is objectionable.

An efficient arrangement inside is to construct a cheap brick furnace with grate bars 2 feet above ground and walls 4 feet high, on top of which may be placed an iron tank, or some old boiler may be set between higher walls. Refuse coal and timber may be used for heating the water, which can be drawn off into tubs; and a water-pipe with a float and faucet can be arranged to keep the water-supply in the tank at the same level.

Racks can be arranged around the furnace for drying clothes, and closets provided around the walls, if needed.

2708. Weighman's Office.—This should be at the tipple, or near the point where the cars are weighed, so that the scale-beam can be erected therein.

If checks are to be issued, and the weights posted daily, provision should be made for proper space around the building, so that the daily weigh sheets can be posted conveniently for the inspection of miners, and check boards arranged for holding a supply of returned checks and new ones for issue.

2709. Mine-Clerk's Office.—If the weighman performs the aforesaid work, a mine clerk may attend to keeping the time of employees, and may have charge of storerooms, supplies, etc., and the issuing of the same, in which case a storekeeper is not needed.

The mine-clerk's office should be located handy to the storeroom and supplies, so that the taking of time can be most readily attended to.

If, however, the mine clerk attends to posting of weights of coal and issuing and receiving checks and keeping of time only, this office will be most conveniently situated near the tipple, if not too distant, or near the mine entrance, so that weigh sheets may be readily inspected by miners in passing in and out of the mine.

2710. Storekeeper's Office.—This is most conveniently situated in the storeroom. If in a separate building, it should be handy to the point of unloading material, supplies, etc., so as to check the same, and convenient to the different storehouses for issuing material to shops and the mines.

2711. Shipping-Clerk's Office.—It may also be necessary to have a shipping-clerk's office near the tipple to attend to shipments of coal, unless the work can be divided as follows:

(1) Weighman to attend to the weighing, posting, and checks, and the mine clerk to attend to time-keeping and shipping; or (2) the weighman to attend to the weighing, and the mine clerk to attend to posting weights, checks, and

shipping, and storekeeper to attend to the supplies and timekeeping.

2712. Mine Office.—An office for the foreman or superintendent, as the organization may require, should be located centrally to the mine operations.

2713. Lamp-Inspector's Office.—At gaseous mines, an office should be provided near the mine opening for the gathering, inspecting, cleaning, and repairing of safety-lamps.

This office should be provided with apparatus for testing lamps and for testing samples of air gathered at points underground, where it is desired to know the amount of gas present, if any. In less fiery mines, the inspection of the lamps may be done by the fire bosses, at points underground, outside of the fire limits. An apparatus should also be at hand to permit of entering gassy places, in case of accident.

2714. Doctor's Office and Hospital.—In operations isolated from settled communities, the above buildings may be necessary, provided with proper surgical instruments, splints, bandages, and medicines, for treating burns, broken bones, and men overcome by carbonic acid gas, after-damp, or other accidents.

The hospital should be provided with cots and other suitable furniture. One or more stretchers should be kept near the mine, for carrying injured men.

2715. Store.—If a store is needed in connection with the operations, it should be located near the railroad-track, if possible, to facilitate the handling of merchandise; it need not necessarily be near the mine, but convenient to the miners' houses.

2716. Miners' Houses.—These should be located at some little distance from the mine buildings. A plot of ground about 50′ × 100′ should be allowed to each house, or, if the space can be allowed, about 300 feet square, so that employees can do some gardening. This should be fenced in; and it may be necessary to lay pipes to the dwellings, to provide a water-supply.

SURFACE ARRANGEMENTS
OF
ANTHRACITE MINES.

DEFINITIONS.

2717. The name **colliery** is given to the entire coal-mine plant. It embraces both the surface improvements and the workings underground.

The term **mine** is applied to the underground workings, shafts, tunnels, and other passageways.

2718. A **shaft** is a vertical opening through the strata which is or may be used for the purpose of ventilation or drainage, or for hoisting men or material in connection with the mining of coal.

2719. A **slope** is an inclined opening used for the same purposes as a shaft.

2720. A **drift** is a horizontal or nearly horizontal passage driven in the coal seam from the surface.

2721. A **tunnel** is a horizontal or nearly horizontal passage driven across the measures.

2722. A **stripping** is an open working, where the soil or débris on top of the seam has been removed preparatory to mining it by an open cut.

2723. The **breaker** is the structure containing the machinery used in the preparation of the coal.

GENERAL PLAN OF ARRANGEMENTS.

2724. The arrangements of the buildings, tracks, etc., at anthracite collieries, or what are generally termed the outside improvements, differ considerably. This is due largely to the topography of the surface.

At a colliery with a shaft opening, the general arrangements differ somewhat from those collieries where the coal is opened by a slope, drift, tunnel, or stripping.

In case of a shaft, the surface is more or less of the same elevation, while for a slope, drift, or tunnel, a sloping surface is more general.

In every case, however, at a well-arranged plant the structures erected upon the surface should all be on the same meridian. In this way the sides and ends of the different structures are arranged parallel to one another. This method, if followed, will aid materially in the construction of maps; besides, it gives a better appearance to the plant than if the different structures are located at random.

At an anthracite colliery there may be a shaft, slope, drift, tunnel, or stripping; or it may be possible that all of these openings exist at the same colliery.

If the opening is a shaft, there is a head-frame; if a slope, provided the slope is on a line with the breaker, there is an inclined plane.

In case the slope is at some distance from the breaker, there is an inclined plane and trestle, an inclined plane and tracks on the surface, or a slope landing; in the last two cases the coal can reach the breaker either over a trestle, or it can be raised by steam-power up a vertical hoist or up an inclined plane.

In case of a drift, tunnel, or stripping, there are tracks leading to the foot of an inclined plane or vertical hoist, or tracks leading to a trestle, where the coal is dumped directly into the breaker without using steam-power to elevate it.

2725. Besides the above, the following are located on the surface:

The *hoisting-engine houses*, with their engines, drums, brakes, etc.

The different *boiler houses*, with their boilers, feeding apparatus, and coal-bins.

The *breaker*, with its machinery for cracking and sizing the coal, and the breaker engine.

The *blacksmith and carpenter shops*.

One or more *fans* used for ventilating the underground workings.

Railroad-tracks.

Offices: General shipping, and for the superintendent and engineers.

Wash-house for miners.

Supply houses: oils, cotton, iron, etc.

Powder house.

Pumping-engine houses, containing pumps to furnish water to wash the coal and supply the boilers.

Water-tanks.

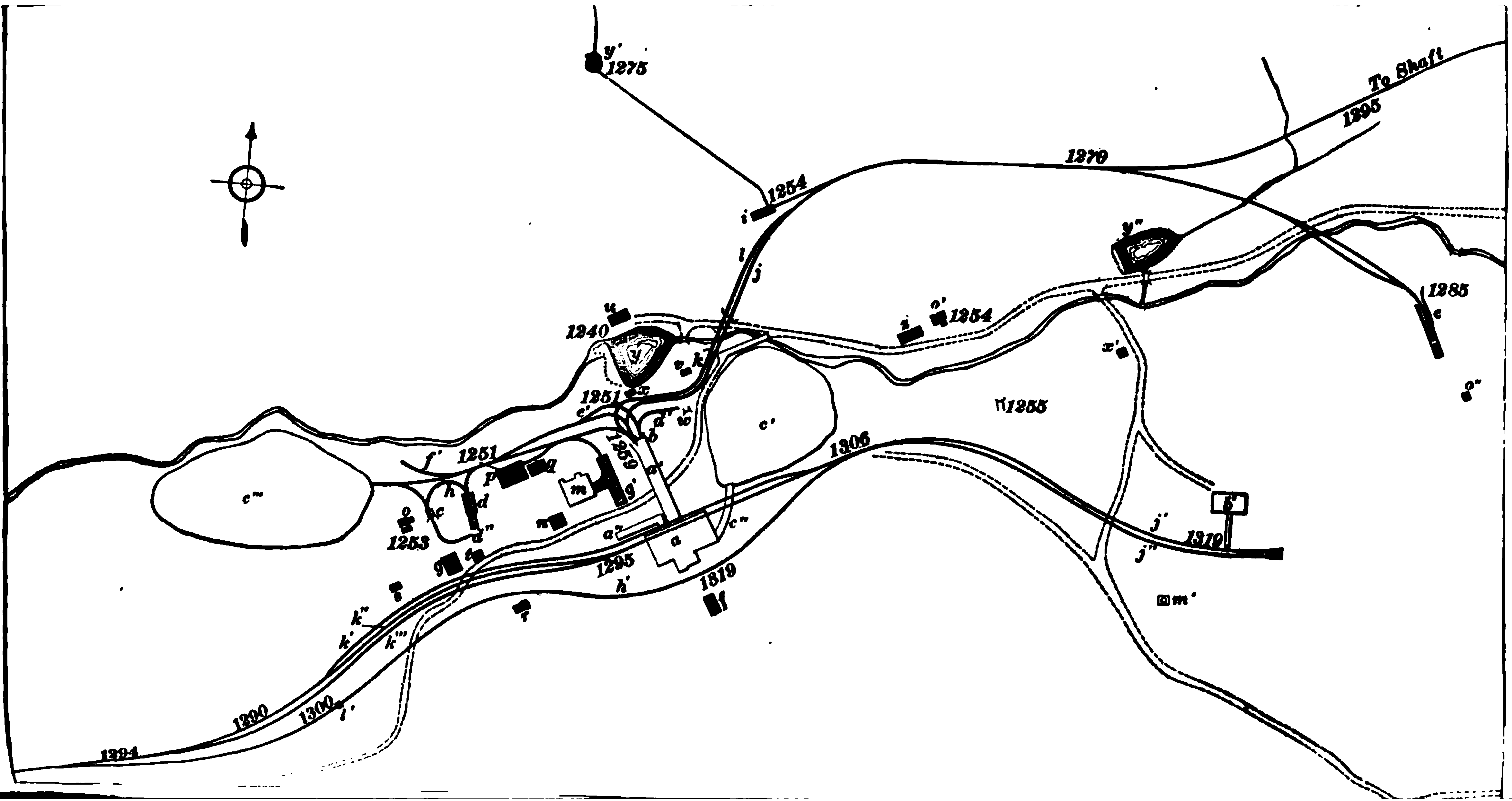
Culm (or dirt, or waste) and *rock banks.*

Different arrangements for getting rid of culm; as by dirt plane, conveyors, trestles, blowers, etc.

2726. Fig. 986 is a plan of the outside improvements at an anthracite colliery, showing the arrangement of the different structures, tracks, dams, etc.

This plan shows that the breaker *a* is fed by one main slope *b*, one tender slope *c*, two tunnels *d* and *e*, and one shaft not shown in this plan, but the tracks leading thereto are shown in part. The coal from the main slope *b* is hoisted up a double-track plane *a'*, leading to the breaker, by means of the main hoisting-engine *f*. The coal from the tender slope *c* is hoisted by the hoisting-engine *g* and allowed to run over a drawbridge. After running over this bridge, it is taken around the loop *h* into the tunnel *d*, in order to have the car ascend plane *a'* with the door in front. The coal from the two tunnels *d* and *e*, the tender slope *c*, and the shaft is all brought to the mouth of the main slope *b* and hoisted up the plane *a'* by means of an engine with friction drum located under the breaker. The plane *a'* contains a single track in connection with the double track used for hoisting out of the main slope.

The grades of the tracks leading to and from the tender slope *c* and tunnel *d* are so arranged that the cars run by gravity. The cars from tunnel *e* and those from the shaft are hauled by a locomotive which is housed in the locomotive


To Shaft
1295
1270
1285
1275
1254
1254
1240
1251
1251
1259
1255
1253
1306
1319
1319
1295
1300
1290
1294

house *i*. The loaded cars are run in on track *j*. The locomotive, being in front of the loaded cars, passes over the cross-over *k* to the empty track, which is elevated above the two loaded tracks at the foot of the plane, and continues on a down grade of 2 to 1 per cent. to the point marked *l*. At the mouth of the tender slope *c* is shown the backswitch, to turn the cars so that they may descend the slope with the door in front.

The surface being a sloping one, the boiler house *m* is so arranged that the ashes are taken out in a small dump-car. The coal used in this boiler house is furnished by a small dump-car running on an overhead trestle, the coal-bins being located within the boiler house. This boiler house furnishes steam for the machinery in the immediate neighborhood, the pumps inside the mine, and heat for neighboring buildings.

There is still another boiler house, to furnish steam for the shaft engines, etc., which is located at a considerable distance from the boiler house *m*.

The water-tank *n* is used to furnish the boilers with the necessary fresh-water supply.

The ventilating fans *o*, *o'*, and *o''* are used to ventilate the different inside workings.

In the blacksmith and carpenter shop *p* the necessary repair work of the colliery is done.

The supply house *q* is used in connection with the blacksmith and carpenter shops, so that the necessary supply of iron, bolts, nails, etc., can always be kept on hand and under cover.

The office *r* is used by the superintendent and shipper, and is so arranged and located that the shipper has a commanding view of the empty and loaded tracks.

The powder house *s* contains every explosive used for blasting in and about the colliery.

At the supply house *t* the miners are supplied with oil, cotton, shovels, and other necessary articles that are used about a colliery.

The engineer's office *u* is where the maps, sections, and different plans used at the colliery are made.

The wash-house *v* is a building where the men can wash and change their clothing. This building is located so that it is within easy reach of the traveling way *w* of the mine.

In the pump-house *x* is a pump which furnishes the water for the different screens used in the sizing of coal. It also furnishes water for the jigs, which are used for separating the slate from the coal, and also the water for the lip screens, over which the coal passes into the railroad-cars for shipment. This pump also furnishes the necessary water in case of fire.

The dam *y* holds the supply of water for the pump located at *x*. This dam receives water from the creek, shown in the plan; but in summer, when the creek supply is small, the water from the mine is pumped into it. In very dry weather, the water from the lip screens is also run back into this dam and used over and over again.

The dam y' furnishes the water for the locomotives; the service-pipe leading from this dam to the locomotive house *i* is also shown.

The water that is used for steam purposes is furnished by the dam y'', the pumping station x' being used in connection with this dam.

All the necessary sawed timber used at the colliery is cut in the steam sawmill *z*.

In connection with the barn b' a small trestle with narrow-gauge track leading to the main railroad-tracks is shown. This track is used to convey the grain from the railroad-cars to the different bins and places of storage in the barn. In the barn a place is set apart for the colliery ambulance, which is used in case of accident about the colliery.

This barn, it will be noticed, is not parallel with the other buildings, but is located to suit the railroad-tracks.

The rock and slate bank c' is located on a steeply sloping side-hill, and the trestle c'' connects the breaker with it.

The slush bank (dirt or waste bank) c''' is where the culm, which is conveyed in troughs by means of water, is deposited.

The tracks *d'* and *d''* lead to the timber-yards, where the timber to be used in the mine is sized and loaded.

The damaged cars are taken off the main slope and conveyed by the track *e'* to the carpenter shop for repairs.

The ashes from the boiler house are conveyed over the track *f'* to the ash-dump.

The pit *g'* is used to hold a mine-car which conveys the lip-coal screenings coming from the lump-coal chute *a''* to the foot of the plane *a'*, where it is hoisted and dumped into the breaker.

The turnout *h'* leading from the main railroad-track is used to run the empty cars over to the empty sidings *j'* and *j''*. From here the empty cars are run under the breaker, where they are loaded and run on to the loaded sidings or tail tracks *k'*, *k''*, and *k'''*, where the loaded cars are allowed to accumulate preparatory to making up a "trip" to be shipped to market.

At *l'* is shown a small opening driven to the surface from the inside workings. This opening is made to dump the condemned coal into, which by means of a chute is loaded into a mine-car, raised to the surface, and again dumped into the breaker and resized and reseparated. This method of handling the condemned coal is a new one, and proves very satisfactory.

From the artesian well *m'* the water is pumped during the dry seasons of the year.

The different figures, or numerals, shown on this plan are the elevations above tide at those points.

TABLE OF SURFACE ARRANGEMENTS.

2727. In the following list the letters reter to Fig. 986.

(*a*) Breaker.
(*b*) Main slope.
(*c*) Tender slope.
(*d*)
(*e*) } Tunnels.
(*a'*) Inclined plane, with three tracks leading to breaker.
(*f*) Main hoisting-engine.

(*g*) Hoisting-engine for tender slope.
(*h*) Loop near tunnel *d*.
(*i*) Locomotive house.
(*j*) Loaded track over bridge.
(*k*) Cross-over on bridge.
(*l*) Empty track.
(*m*) Boiler house.
(*n*) Water-tank.
(*o*), (*o'*), (*o''*) Ventilating fans.
(*p*) Blacksmith and carpenter shop.
(*q*) Supply house to blacksmith shop.
(*r*) Superintendent and shipper's offices.
(*s*) Powder house.
(*t*) Supply house: oil, cotton, etc.
(*u*) Engineer's office.
(*v*) Wash-house.
(*w*) Traveling way.
(*x*) Pump-house.
(*y*) Dam for breaker.
(*y'*) Dam for locomotive.
(*y''*) Dam for steam supply.
(*x'*) Pumping station for steam supply.
(*z*) Sawmill.
(*b'*) Barn.
(*c'*) Rock and slate bank.
(*c''*) Trestle leading to rock and slate bank.
(*c'''*) Slush bank.
(*d'*), (*d''*) Tracks leading to timber-yards.
(*e'*) Track to take cars off main slope.
(*f'*) Ash-dump track.
(*g'*) Pit for lump-coal screenings.
(*a''*) Lump-coal chute.
(*h'*) Main turnout.
(*j'*), (*j''*) Empty turnouts.

(k')
(k'') Loaded turnouts.
(k''')
(l') Condemned-coal chute.
(m') Artesian well.

THE DESIGN OF A PLANT.

BREAKERS.

2728. In the designing of a plant, great care and judgment must be exercised. The designer must look to the future of the plant and not merely to its immediate requirements, unless only a temporary structure, commonly known as a **coffee-mill,** or **penitentiary,** is required.

These structures are merely small breakers, used where overlying veins are worked that can not be mined from the main openings, or where pillars are robbed that have been left intact around the main openings, on account of some cave-in of the main haulage roads. They are also used to supply the demand for coal for domestic purposes, or in some out-of-the-way place where the farmers are supplied with fuel.

After the main openings for the colliery have been decided upon, very rude temporary structures are erected merely to do the work of "opening up," as it is termed. These temporary structures are so erected as not to interfere with the subsequent erection of the permanent ones.

The first thing for the engineer to do is to run section lines and locate all outcrops and streams. Sections are then constructed and a temporary map made.

A meridian is next selected; if the opening is a slope, the slope line is frequently taken, which is the line on which the slope is driven. If the opening is a shaft, a center line of one of the hoistways is frequently taken; generally this line is in the direction of the main haulage ways at the foot of the shaft.

After the meridian has been selected, all the buildings, or as many as practicable, are erected with their sides parallel to the meridian, so as to secure uniformity.

In every case the plant must be as compact as possible, and at the same time comply with all the laws that are laid down for its construction.

2729. Site for Breaker.—The first thing to consider, after the opening or openings have been decided upon, is the site for the breaker. This, in every case, will depend upon the mine opening or openings and the topography of the country. Where the topography of the surface permits, the location of the breaker should be at a point low enough to bring the top of the breaker below the level of the mouth of the shaft, slope, drift, tunnel, or stripping, so that a descending grade can be obtained from the opening to where the cars are dumped. Such a location is preferable to all others, but it can not always be obtained.

In some cases it is found better to locate the breaker in line with the slope, so that the cars can be hoisted and dumped directly into the breaker.

In many places where gravity can not be used, or where hoisting directly into the breaker is impossible, a rope or chain system is used, or, frequently, a locomotive or an electric motor conveys the coal from the opening to the location of the breaker.

One of the main points to be taken into consideration, in choosing a site for a breaker, is the location of the shipping tracks, for in all cases the breaker is so located that railroad-tracks can be constructed with sidings having grade sufficient to move the cars by gravity.

The location of the breaker being decided upon, the excavations for the masonry foundations are immediately pushed forward.

2730. The Masonry for a Breaker.—This is a class of work that is a matter of choice; some prefer a continuous wall, where a sill is used, as in Fig. 987, where *a* shows a side view and *b* an end view of the wall, with sill and post in place.

Others prefer a pier *c* and capstone *d*, as shown in Fig. 987. In this case the post rests either directly on the cap-

stone or, as is often the case, a small piece of No. 8 sheet iron *e* is placed between the post and capstone. Or, a cast-iron shoe may be used instead of the sheet iron. Still others prefer a continuous wall- *f*, with capstones *g*, *g'* for the posts, also

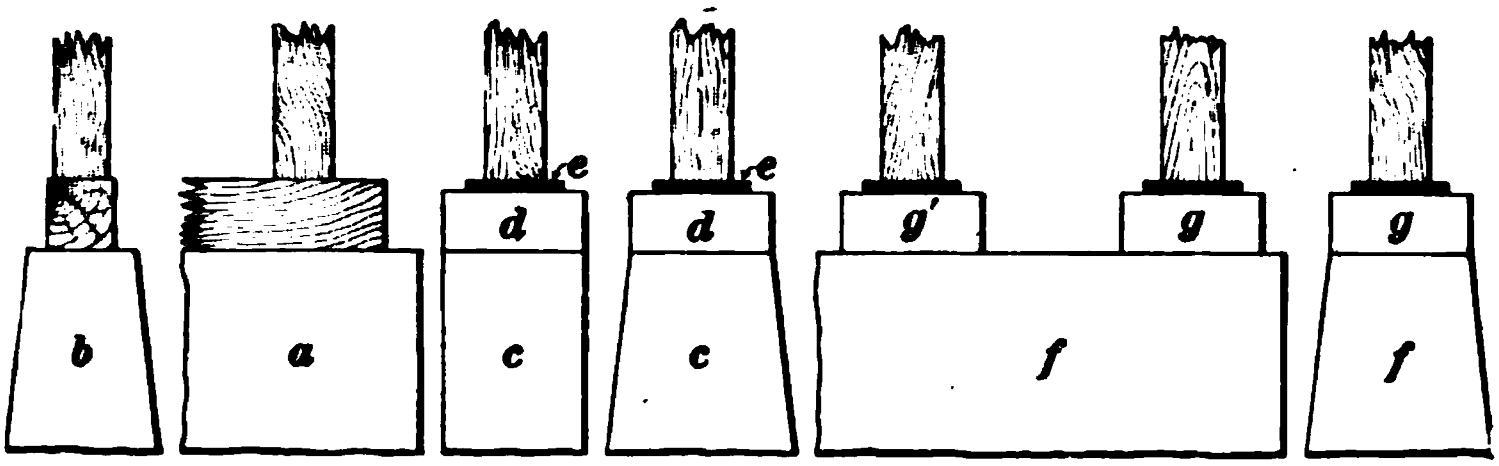


FIG. 987.

as shown in Fig. 987. As in the above case, either a piece of sheet iron is used or a cast-iron shoe is placed between the post and capstone. These walls are generally hammer-dressed, set in mortar or cement, while the capstones are all tool-dressed.

STEAM PLANT.

2731. The breaker location being determined upon, the next site to be selected is for the **steam plant.** The choice of location is influenced by the Anthracite Mine Law of Pennsylvania, which specifies that "it shall not be lawful to place any boiler or boilers for the purpose of generating steam under nor nearer than one hundred (100) feet to any coal-breaker or other structure in which persons are employed in the preparation of coal."

The location of the steam plant for the breaker and the immediate hoisting and pumping engines should be such that the arrangement for supplying the plant with fuel is as simple and inexpensive as possible. Very often the topography of the surface permits the erection of a chute, through which the coal is run direct from the breaker to the plant, and distributed by what are called **telegraphs.** At times it is very convenient to put up a small pocket for the boiler supply, and convey the coal to the boilers by means of an overhead trestle and small dumper.

In another case, it may be convenient to put up a system of conveyors. In every instance, two main objects must be kept in view: First, that of supplying the steam plant with fuel; second, that of having the plant located as centrally as possible, so that the steam, in traveling to the different places of usage, will be subjected to as little condensation as possible.

One great point is to have the steam plant all under one cover, and not to have one set of boilers for the breaker at one place and a set for the hoisting-engines at another.

Of course, where the openings are at a very great distance from the main structure, it is not to be expected that all the boilers can be under the same cover.

2732. The handling of the ashes from a steam plant is always a secondary consideration. Frequently, the surface will permit of the erection of a plant where a pit *a*, as shown in Fig. 988, can be dug directly in front of the boilers, so that a small dump-car can be used to convey the ashes to the ash-dump. Sometimes a line of conveyors is put in to handle the ashes in this pit.

In case a pit is constructed and a dump-car used, the pit should be open at both ends, otherwise there is a possibility of gases accumulating in the pit. A case is on record where the pit was closed at one end and deadly gases accumulated. The continued absence of the man removing the ashes was noticed, and, upon searching for him, he was found dead in the pit from having inhaled the gases. A method sometimes employed in removing the ashes is by a small dump-car running on a track located directly in front of the fire-box.

Most frequently a cart and mule or a man and wheelbarrow are used.

2733. The steam-boilers in use at the present day are so numerous that the engineer or person in charge will have no trouble in making a selection to suit his wants. The old style of cylindrical boilers is giving place to a style that is better adapted for the mines of the anthracite region.

The Stirling water-tube boiler, the Babcock & Wilcox water-tube, the National water-tube boiler, and others of a similar type are in numerous instances replacing the old

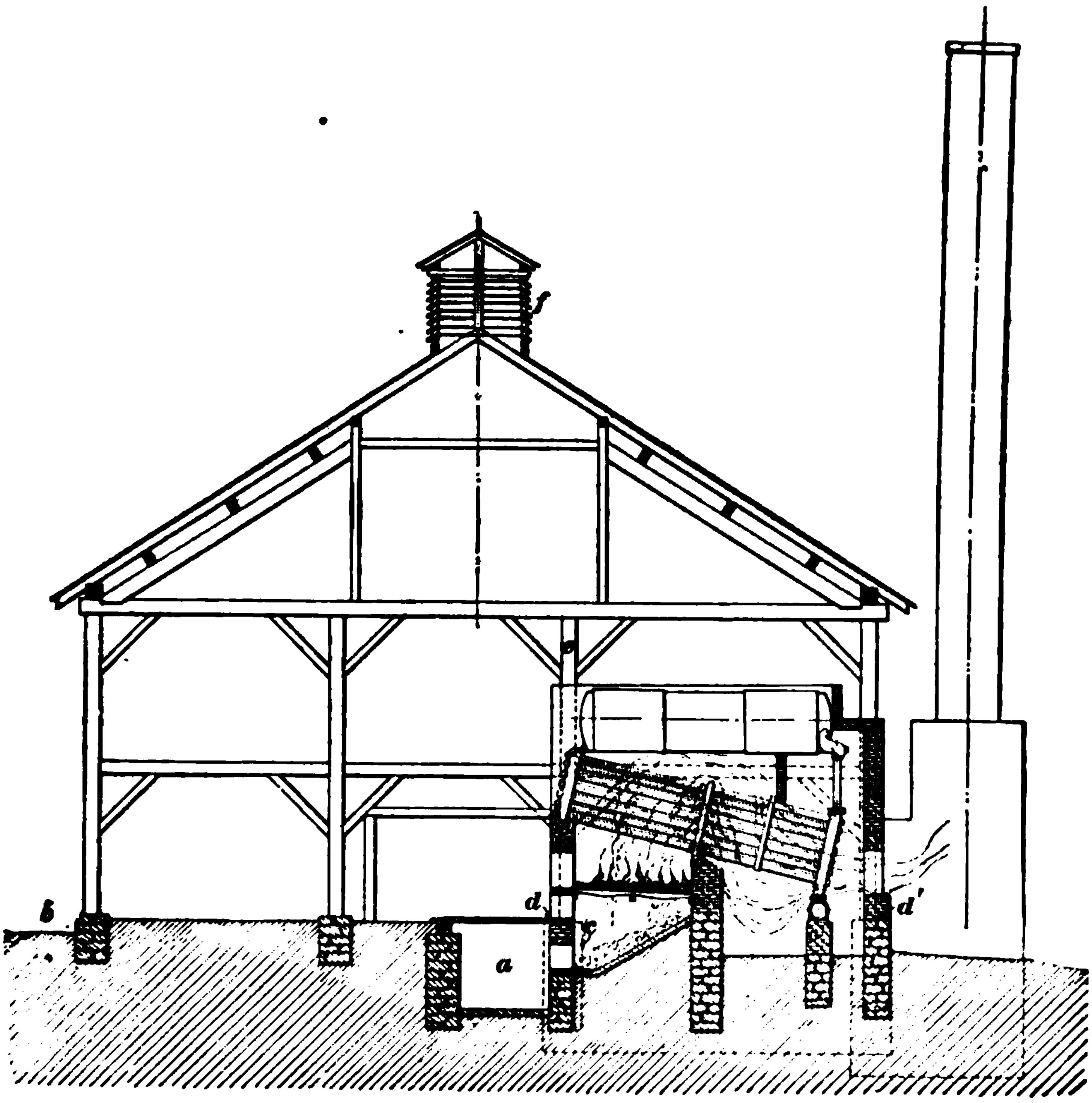


FIG. 988.

style of cylinder boilers. They are better steam-generators, and are more efficient for the same amount of coal used.

2734. It matters not what boiler is decided upon, the foundation for the same should be very substantial. In excavating, a good solid bottom must be secured, so that in after years no trouble will arise from the walls settling.

The stonework upon which brick walls are erected to enclose the boilers should be brought up out of the ground at least to such a height that the wall will be on the same level as the front piers supporting the boiler house, as shown

at *b* and *c* in Fig. 988. If this method is followed, scarcely any trouble will be experienced when the boiler-house floor is laid.

Generally, the stonework is simply a hammer-dressed wall. The brickwork is generally set back from the face of the stonework from 6 to 8 inches, as shown at *d* and *d'*, Fig. 988.

BOILER HOUSE.

2735. The boiler houses, as commonly built, are frame structures. The rudest kind of a shed answers the purpose at some collieries, but quite substantial stone and brick houses are seen at others.

Fig. 988 shows the side elevation of a framing for a boiler house often used in the anthracite region. The sheathing most commonly used is 1-inch white pine or hemlock boards or sheets of corrugated iron.

The covering for the roof may be either shingles, corrugated iron, or slate. In case a shingle or corrugated-iron roof is put on, it should be coated with a good covering of mineral paint. Some prefer a slate roof to all others for boiler houses. They complain of the shingles warping on account of the steam, and the iron corroding in case both sides are not kept well covered with a coat of paint.

In the designing of a boiler house, the members of the different trusses must be made high enough above the boilers so that they do not interfere with the erection of the steam connections.

Another point to have in view is the location of the doors, so as to give the men in charge of the boilers the benefit of any breeze in summer-time. To do this, the post *e* holding the door should be set back a little from the face of the boilers. The boiler house should be constructed so as to give ample room in front of the boilers for a supply of coal that will last several days. Every boiler house should have a ventilator *f*, to allow the steam and gases to escape. The floor of the boiler house should be laid so that it slopes towards the boiler.

2736. It is economical to have a feed-water heater located in the boiler house or at some place near by, so that the water before entering the boilers can be heated. There are various forms of feed-water heaters in use; one that is very simple in construction is a cylindrical boiler fitted up for the purpose. The water is heated by running the exhaust-steam into it from the breaker engine, the hoisting-engine, a pumping-engine, or from the fan-engine. It is also arranged so that a jet of steam can be used at any time, as at night, when the engines are not in use.

2737. In the anthracite regions at the present day much attention is paid to the different methods that utilize the finer sizes of coal and culm for steam purposes. The most improved system of grates and blowing apparatus is used, whereby the finer sizes of coal, and very often culm, can be burned.

Where culm is used, a great deal of hard manual labor is required. To overcome this, a mechanical stoker is sometimes employed, which can be easily attached to almost any stationary boiler in use at the present time. With this arrangement, the manual labor is reduced to a minimum.

HOISTING-ENGINES.

2738. The location of the different hoisting-engines used about a colliery depends upon the kind of opening, whether it is a shaft or slope, upon the topography of the surface, and the location of the opening, whether it is in connection with the breaker or at some distance from it.

With a shaft opening, the distance between the hoisting-engine and head-frame should be such that the rope will coil regularly on the drum. It should also be located so that it will not be necessary to put carrying pulleys between the drum and head-frame to overcome the violent oscillation of the rope that results from an improper location.

2739. In case of a shaft, an arrangement called a **vertical hoist** is employed, except where the coal is conveyed to the top of the breaker by means of tracks or chute,

or, as is very often the case, by an **inclined hoist.** For the Anthracite Mine Law of Pennsylvania specifies that "no inflammable structure, other than a frame to sustain pulleys or sheaves, shall be erected over the entrance of any opening connecting the surface with the underground workings of any mine, and no breaker or other inflammable structure or structures for the preparation or storage of coal shall be erected nearer than 200 feet to any such opening." In case of a vertical hoist, the winding-engine used to operate this hoist is generally located in the lower part of the breaker, or the breaker engine may be used, the drum being provided with a friction-clutch.

At collieries where the coal is raised through a slope, and the slope is connected with the main structure by an inclined plane, the location of the engine, if the topography will permit, is on line with the slope at some point back of the breaker. On a side-hill this location is preferable to all others.

2740. Where hoisting is done over the breaker, the winding-engine is sometimes located within the lower part of the breaker. This is very poor practice, and should be avoided, for a number of breakers have been destroyed by fire, the origin of which was directly or indirectly traceable to the engine room. Again, the rope passing through the breaker on its way to the drum is an annoyance, and often interferes with the erection of improvements that are desirable in the breaker after it has been in operation for some time.

2741. In case of slopes that are located some distance from the main structure, the drum for the winding-engine or engines should be located one hundred and fifty (150) to two hundred (200) feet from the knuckle, so as to secure a sufficient distance between the knuckle pulleys and the drum, that the rope may coil regularly on the drum. In case the drum is above the level of the tracks, the height in connection with the distance should be such as not to interfere with the hitching and unhitching of the car, and at

the same time give ample room for the arrangement of the empty and loaded tracks.

When the winding-engine and drum are placed below the level of the slope knuckle, the rope is sometimes passed through a slide, twelve to eighteen feet long, working like a cross-head between guides. The slide is attached to a counterweight, and the hole through which the rope plays, although large enough to pass the rope freely, will not pass the rope-socket, the hook, or a stop placed on the chain. After the loaded car is detached, the drum is turned back one-quarter or one-third of a revolution; the counterweight upon the rope keeps it tight upon the drum, and pulls it out to within a few feet of the knuckles, where the empty car is to be attached.

In many cases where the winding-engine is below the slope knuckle, the rope coming from the winding drum is passed over a sheave wheel set on a frame, which is built high enough to bring the rope above the level of the tracks at the head of the slope.

2742. At many of the anthracite collieries hoisting-engines are located on the surface to operate inside slopes and shafts. Where such engines are in operation the rope is led from the drum into the mine through a bore-hole, or in many cases it is conveyed through an old breast that is worked to the surface, or through a traveling way, pump way, airway, or an air-shaft opening.

These engines are erected upon the surface, in many cases, to avoid the damaging effects that the steam has on the inside workings when they are located underground; besides, there is then no loss of steam by condensation from conveying it a great distance into the mine. Where these engines are in use, they usually are set from 100 to 150 feet from the bore-hole.

Instead of erecting these engines on the surface to overcome the disadvantage of exhausting into an airway, they are sometimes put down in the mine, and the exhaust from the engines is led into a good-sized bore-hole that has been

lined to keep out the water from the strata it passes through. Bore-holes are also used for conducting the steam into the inside workings from the surface.

2743. Engines that are used to lower and hoist men into and out of the mine (these are generally the main winding-engines) should be separated from all other engines or machinery and from the sound of gongs used for signal purposes for other machinery, so that the attention of the man in charge of the engine will not be drawn away from his work.

2744. Anthracite collieries have all types and classes of engines. The engines used for hoisting purposes are of the horizontal high-pressure type, either direct-acting or geared, single or double.

Where a large number of mine-cars must be handled daily, powerful double direct-acting engines are necessary.

For single-track slopes, engines which have drums loose on the shafts are used. These are prevented from revolving by means of a friction-clutch.

The friction-clutch can be applied to such a drum with equal facility, whether the engine is at rest or in motion. When this arrangement first came into use, the friction-clutch was applied by means of a hand or foot lever, but at present it is applied by steam. The improved device is constructed so as to gradually engage the drum as it is thrown into gear, thus avoiding shock or strain on any part of the engine.

The load is lowered by means of a powerful hand or steam brake, or, at the option of the operator, by the friction itself.

2745. The principal requirements of anthracite winding-engines are:

1. The engine must be thoroughly under the control of the engineer, so that it can (*a*) be quickly stopped when running at full speed, and (*b*) be moved with certainty and nicety through a small fraction of a revolution. This is necessary in landing.

2. It must be capable of being quickly started with full load at any part of the stroke.

3. It must be capable of attaining full speed in two or three revolutions.

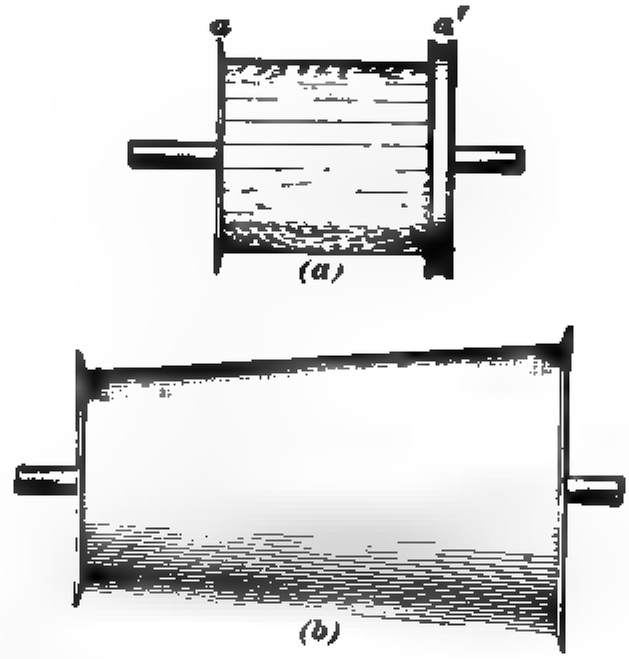
4. Great strength in every part is absolutely essential to prevent breakage from the severe shocks to which winding-engines are always subjected.

5. Its construction must be as simple as possible.

6. Every part of the engine and drum should be easily accessible, to facilitate repairs.

2746. Drums.—The drums in use for hoisting-engines are of different types, and are seldom less than 6 feet in diameter.

Cylindrical drums are in more general use than any other type, except for very large shaft engines, where the double conical drum is generally used.



(a)

(b)

2747. In Fig. 989, (*a*), (*b*), and (*c*) show the types of drums generally used at collieries in the anthracite region; (*a*) and (*b*) are for slope engines, and the drum (*c*) for shaft engines.

The type shown at (*a*) is usually constructed with 2 or 3 spiders having 6 or 8 arms, surrounded by a lagging of timber from 6 to 8 inches thick. The part *a* is one of the flanges or horns that are used to comply with Section 15 of the Anthracite Mine Law of Pennsylvania. These flanges are sometimes

cast in one piece with the drum spider, or, as is very often the case, the flanges are cast separately and bolted to the drum.

The place for the brake band is shown at *a'*. This is sometimes cast in connection with the spider, especially where a steam brake is used.

When the part *a'* on which the brake band is closed is not cast, blocks from 6 to 8 inches thick are bolted to the lagging of the drum, or the brake is applied directly to the drum laggings. This, of course, is a very poor practice.

The drum (*b*) is a cast-iron one, and is in use at but few collieries in the anthracite regions.

The double conical drum (*c*) generally has an attachment on its inner side to adjust the rope. This is of great benefit for shafts, for after the rope has been in use for some time one end or the other needs adjusting.

When the engine is not direct-acting, the spur-wheel used for driving the drum is often placed in the center of the drum, instead of at either end. This has been considered one of the best plans for geared engines, as it reduces and localizes the torsional strain to which the drum shaft is subjected.

2748. Brakes.—A great number of different types of drum brakes are in use at the present time. The old style of brake-blocks are fast giving way to the iron band, which is now in general use throughout the anthracite region. This band is operated by hand or steam-power. In case of a hand-lever, the force is multiplied by using several short levers. Where steam is used to apply the force against the brake-lever, the brake is generally termed a **steam-brake.**

2749. Steam-brakes for hoisting and haulage engines have always been considered very desirable, but trouble was met with in their use on account of the sudden jumping and irregular movement of the piston, and the shock to the cylinder and its connected mechanism at the end of the stroke. This appears to have been overcome in the design of the Zehnder steam-brake, shown in Fig. 990. This consists of two cylinders, the upper one being an air-cylinder

with a port controlled by a valve of peculiar construction near each end. The lower cylinder is a steam-cylinder. Both cylinders are supplied with a piston and piston-rod, connected with each other by a cross-head. On the cross-head connecting the two piston-rods is a pin which is connected with the lever leading to and working the brake device on the drum. The air-cylinder, which receives its air through the valves located on the top, forms an air-cushion which prevents the steam piston striking against

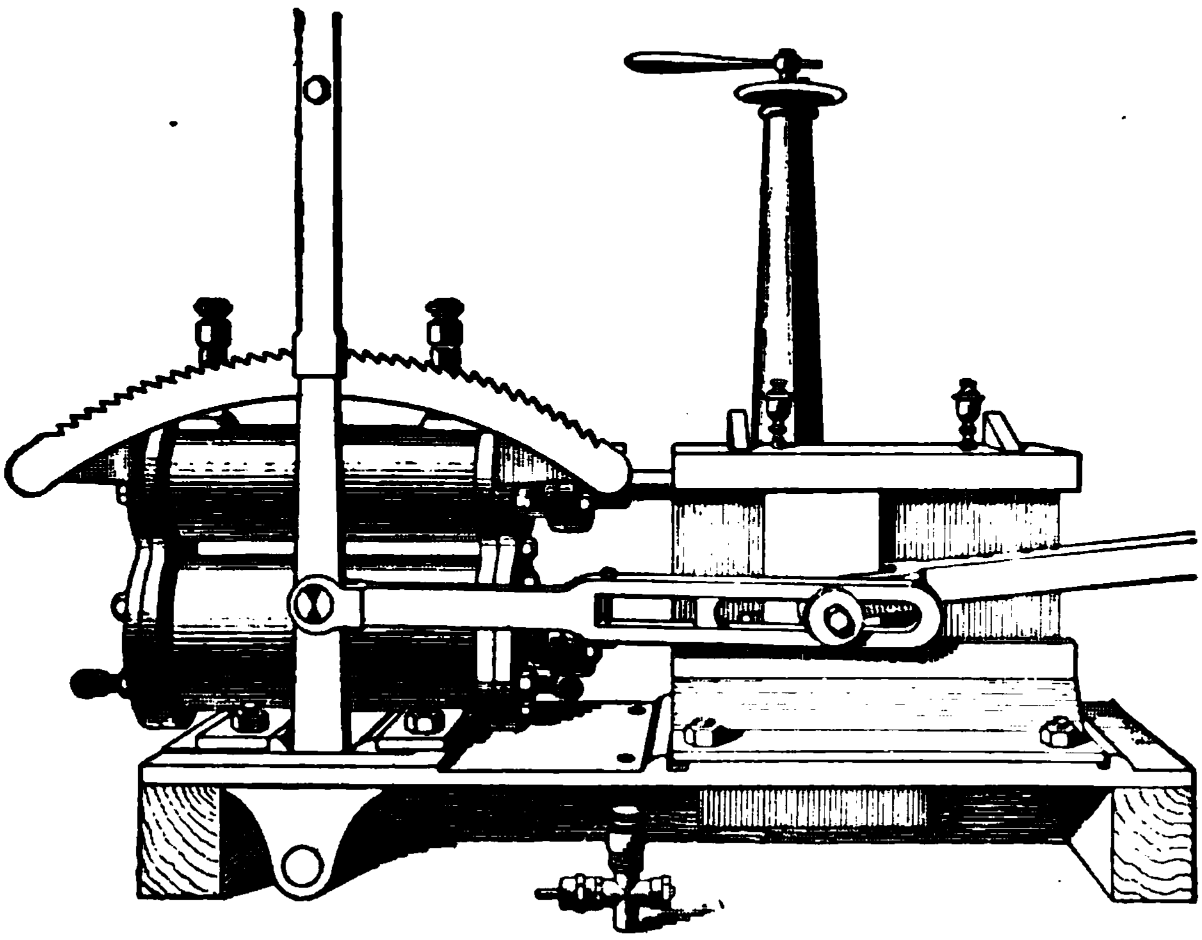


FIG. 990.

the end of the steam-cylinder; a hand-operating lever is attached to the power brake in such a manner that it can be used to set the brake in cases of emergency, and at the same time not interfere with the movements of the steam mechanism.

BREAKER ENGINE.

2750. The breaker engine, which is the engine used to drive the machinery connected with the breaker in the preparation of coal, is generally located in the lower part of the breaker.

There are breakers where the engine is located some

distance away, so as to guard as much as possible against fire.

Where the engine-house is separated from the main structure, a wire rope is used for transmitting the power.

In the latest construction of anthracite breakers, the breaker engine is placed within the main structure. This would seem to indicate the proper place for the engine, for the power will be more direct, and in case of accident the man in charge of the engine can oftentimes discover it before he is signaled.

The breaker engines are generally of the horizontal type, and both single and double ones are in use. Their size will depend upon the amount of machinery in use.

Very often engines are located in different parts of the breaker to run a special piece of machinery, and sometimes a separate engine is used to run each set of jigs.

By the use of these different engines a great deal of work is taken away from the main breaker engine. However, it is better practice to operate everything in the breaker, with the exception of the jigs, by one engine.

A breaker engine must always be powerful enough to supply extra power in case it is necessary to put in improvements after the breaker has been in operation for some time.

The size of hoisting-engines can be very readily computed, but a breaker engine is generally selected by comparison with others. If a 125-horsepower engine is in use at a colliery preparing 1,500 tons of coal per day, and the new plant has coal of about the same grade, needs about the same amount of machinery to prepare the coal, and carries the same pressure of steam through the same length of pipes from boilers to engine, a good basis is at hand for making the selection.

Every breaker engine should be fitted up with a governor, so the speed of the engine may be regulated. It should also have a self-acting lubricator, one that can be set so that the cylinder can receive a sufficient quantity of the lubricant. The different journals of the engine should also be supplied with self-oiling cups, for in most breaker-engine rooms there

is more or less dust, and this, with the continual running of the engine, requires the journals to be well supplied with oil.

2751. Indicating Engines.—Some coal companies in the anthracite region pay much attention to the indicating of engines. Economy of steam is their prime motive, and to attain it the old style of slide-valve engine is giving place to those special designs which admit of a greater number of expansions than can be obtained with the ordinary slide-valve. Such valves are better adapted to breaker and fan engines, which run continuously, than to hoisting-engines, where there is a continual starting and stopping.

2752. Engine Foundations.—A great deal of attention is paid to the engine foundations at the present day. Almost without exception they are built of stone. The stone of which the foundations are constructed may be of any kind, so long as it is durable. Most mining properties have an abundance of conglomerate rock, which will answer very well for engine foundations.

In excavating, a good solid bottom is sought, and very often masonry is built on the bed-rock. The general practice is to build the foundation-bolts in the masonry.

Wooden templets are made at the shops where the engines are manufactured, and sent to the colliery. They are set up on some framing erected about the excavation. Some line on the templet is assumed in connection with the lines of the slope or shaft. The templet is leveled up and put in position by an experienced mechanic.

In finishing, the top of the foundation is made as nearly level as possible. After the bed-plate of the engine has been set in position, it is leveled up by introducing small iron wedges between the bed-plate and the top of the foundation. After the foundation-bolts have been securely drawn up, sulphur is used to fill up the small openings existing between the bed-plate of the engine and the top of the foundation, so as to give the engine a solid bearing throughout.

The height to which the foundations are brought up is

governed very often by the topography of the surface, but mostly by the amount of clearance needed by the drums, fly-wheels, and belt wheels that are in use, and also by the amount of stone necessary to insure absolute stability.

A distance of 8 to 12 inches is not too much space to allow between the edge of the bed-plate and the sides of the foundation.

DRAINAGE AND PUMPING MACHINERY.

2753. The pumping machinery used about an anthracite mine is for draining the mine, supplying the boilers with water, supplying the different washing apparatus in and about the breaker, or for use in case of fire.

In mine drainage, the water that can be caught by a water-level gangway, opening by drift or tunnel to the surface, is conveyed by gravity directly from the mine.

The water in a mine below water level is either pumped out by a pump located on the surface or in the mine, or it is hoisted by the winding-engines in a special car, known as the **water car.**

2754. There are two classes of pumps in common use in the anthracite region for draining mines—the *outside plunger-pumps* and the *inside steam-pumps.*

The first class of pumps is arranged on the surface directly over or in line with an opening called the **pump way,** down which the pump-rods and column pipes are carried into the mine. The class of outside pumps which meets with most favor at the present time in the anthracite region is the **bull-pump.**

Steam-pumps of almost all forms, sizes, and makes are to be found inside the mines throughout the anthracite region. The number of collieries at which no steam-pumps are used is comparatively small.

The inside steam-pumps are nearly all plunger-pumps, as they are more suitable for the strongly acidulated mine water than the ordinary piston pumps.

The Goyne, Jeanesville, Stockton, Allison, Worthington, Cameron, Knowles, and Laidlaw-Dunn-Gordon are some of

the steam-pumps most commonly used in the anthracite region.

At some of the mines, instead of using steam to operate the inside pumping machinery, compressed air or electricity is employed. Where this is done, the generating plants are erected on the surface at some convenient place near the mine opening.

2755. At collieries not provided with pumping machinery, the water is raised by the winding machinery in a water car.

At slope collieries this car generally consists of an old cylindrical boiler-shell provided with large flap-valves, and mounted on the bottom framing or truck of a mine-car. When coal is not being hoisted, the water car is run on the slope and the water raised. The valves are so arranged that when the car is lowered into the water the boiler is filled, and on reaching the surface it can be quickly emptied by means of a lever opening the valves.

At shaft collieries the water car is of special design, and is generally attached directly to the rope, the cage being detached during the operation of hoisting water. These water cars or tanks are also filled at the bottom of the shaft by a valve arrangement in the bottom of the car, and are emptied by an automatic arrangement at the surface. The pumps for outside purposes are similar to those mentioned in the list of steam-pumps for use inside the mine, but of a smaller pattern.

HEAD-FRAMES.

2756. Having decided upon the location for the hoisting-engine, the erection of the head-frame is begun, in case the opening is a shaft.

The head-frame is a support for the sheaves, or wheels, over which the winding-ropes are led from the drum, which is located at a short distance from the shaft. At some of the colleries in the anthracite region iron and steel head-frames are being erected. These are far superior to the

timber head-frames for shafts, on account of their durability and their indestructibility by fire.

There are places in the anthracite region where a steel or iron chute is used, in connection with a steel or iron head-frame, to convey the coal to the breaker, instead of conveying it by tracks over the surface.

2757. In the anthracite region there are three classes of head-frames in use :

1. The triangular form.
2. The square upright pattern, with or without inclined braces.
3. An upright frame, with inclined braces.

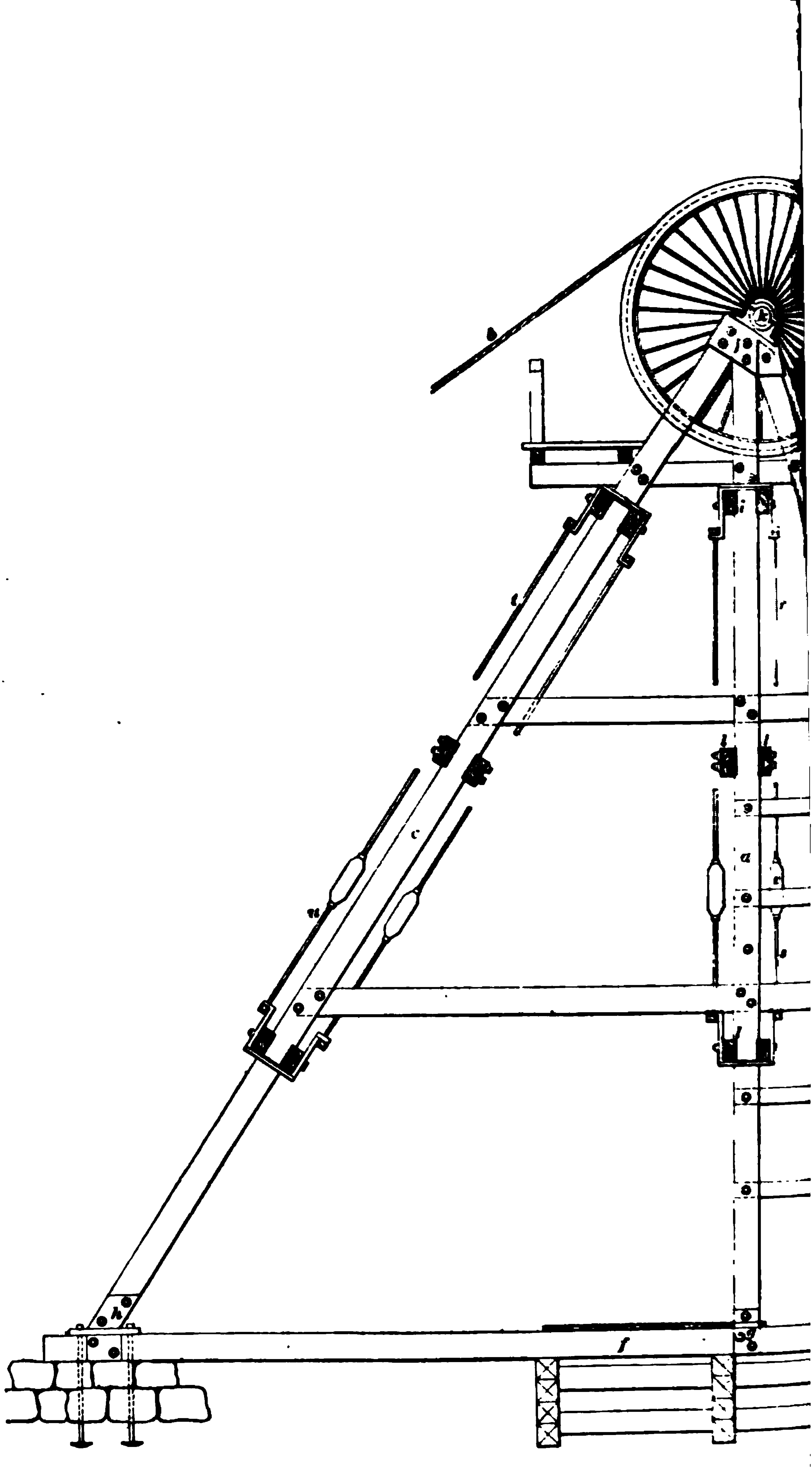
2758. Fig. 991 shows the construction of a triangular form of a timber head-frame which is largely used. In this figure are shown a side elevation and an end view taken directly in front of the upright post marked *a* in the side view.

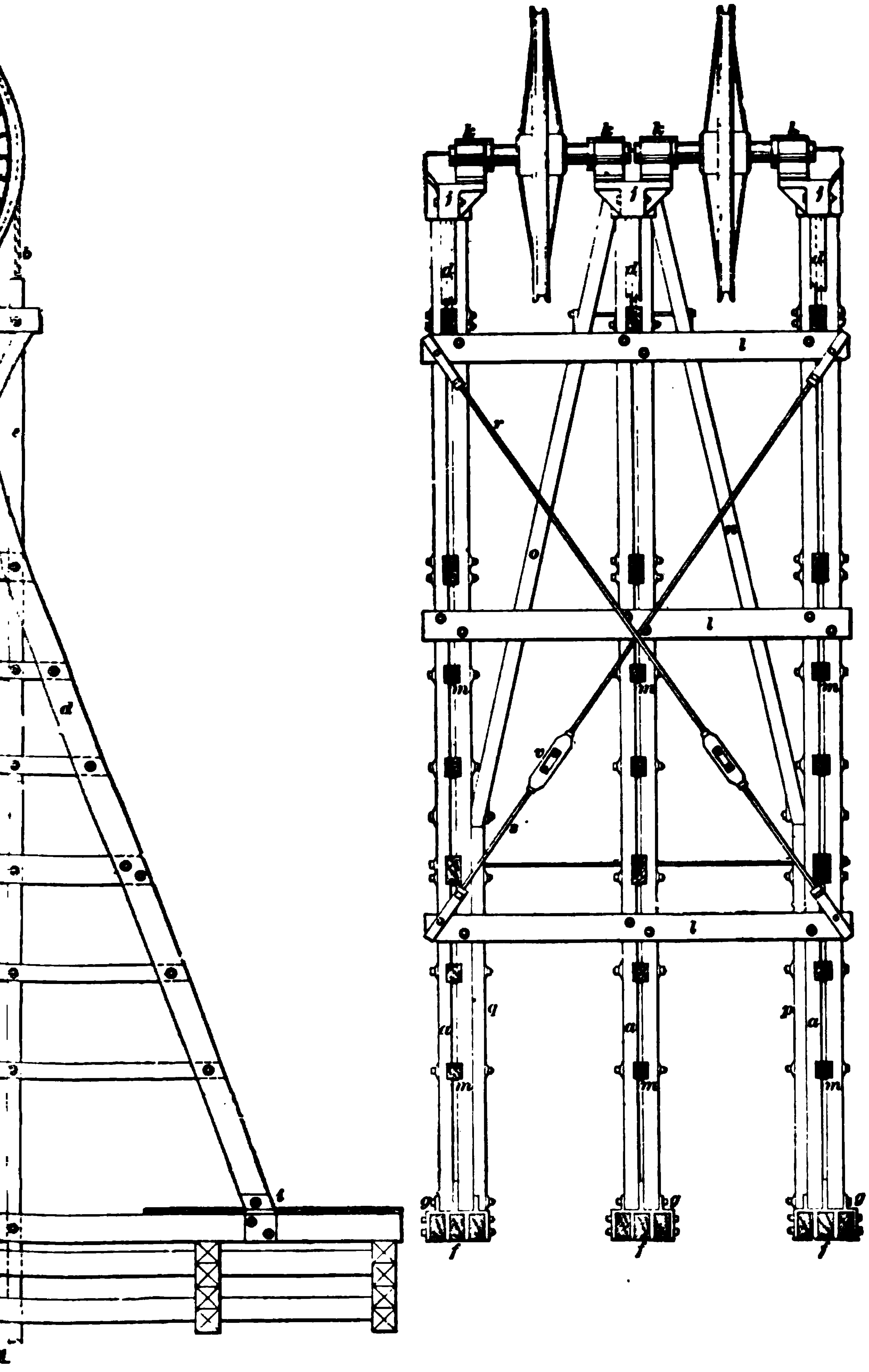
In the construction of the above form of head-frame the location of the drum and of the shaft are known; the height of the head-frame is then decided upon, and is usually made from 30 to 50 feet. With direct-acting engines this height should be sufficient to allow a play of at least two-thirds of a revolution between the cage landing and the overwinding point.

2759. As shown in Fig. 992, S is the sheave and D the drum. The two forces $a\ D$ and $a'\ D$ act towards the drum, and two vertical forces act down the shaft approximately equal to the two forces acting towards the drum. There are, therefore, two resultants, $a\ b$ and $a'\ b'$, the directions of which are determined by lines from a and a' through the center of the sheave S.

This diagram shows that the structure of maximum stability will have a vertical limb parallel to the vertical forces, and an inclined limb approximately parallel to a line joining the centers S and D; but as it is not usually feasible to make $A\ S$ parallel to $S\ D$, the inclined limb is given less inclination.

Theoretically, the brace $A\ S$ should be in the direction of


c
a
h
f



k
j
d
l
r
o
m
v
s
a
p
q
g
f
b
c
t

and parallel to the resultant, but at times the structure is subjected to variable strains in hoisting; consequently, the

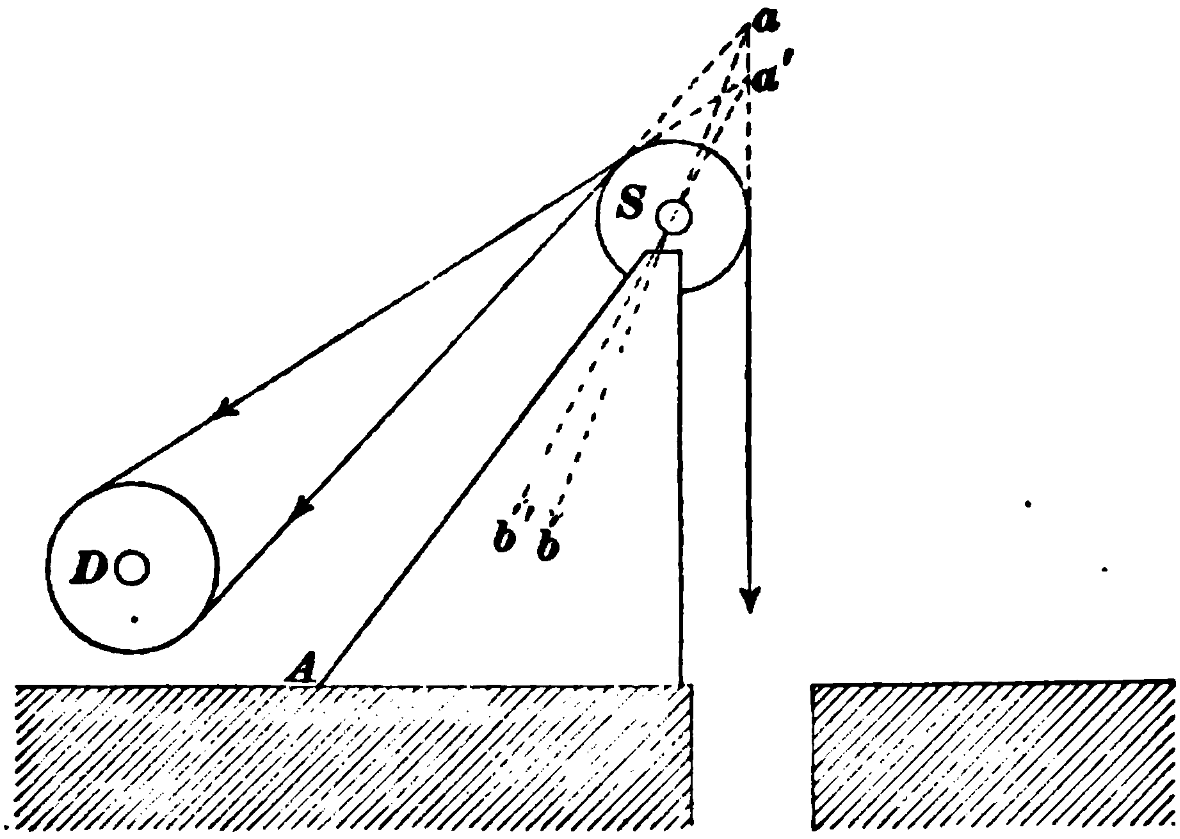


FIG. 992.

direction of the brace *A S* will be somewhere between the resultant and the line of the under-winding rope of the drum.

2760. Again referring to Fig. 991, the construction shows the head-frame to be made up of the posts *a*, which are parallel to the winding-rope *b* running down the shaft; the inclined brace *c*, which resists any thrust that would tend to rotate the head-frame; the inclined brace *d*, to which are secured the cross-timbers *m* that support the cage-guides *e*.

As shown in this figure, the sills *f* are made up of three pieces of timber 8 inches × 14 inches in cross-section. The posts *a* rest in cast-iron shoes *g*; the shoes, as shown, are firmly bolted to the posts and sills. The inclined braces *c* and *d* are fitted with cast-iron shoes *h* and *i*.

Where the post *a* and the two braces *c* and *d* unite at the top of the frame, they are held in place by the casting *j* which supports the pillow-block *k*.

The posts *a* and the brace *c* are made up of two pieces of timber 8 inches × 14 inches in cross-section. The brace *d* consists of one piece of timber 8 inches × 14 inches in cross-section. The transverse timbers *l*, which are used for

bracing, are two pieces of timber 6 inches × 14 inches in cross-section.

The timbers *m* supporting the guides are single pieces of timber 8 inches × 8 inches in cross-section.

The center post, as shown in the cross-section, is braced by the two pieces *n* and *o*, which are supported by the two timbers *p* and *q* bolted to the two outside upright posts. The upright posts *a* and the inclined brace *c* are further braced by the tie-rods *r*, *s*, *t*, and *u*, all of which are fitted with turnbuckles, as shown at *v*. The different posts are firmly bolted together, the bolts being fitted with cast-iron washers.

INCLINED PLANES.

2761. When the opening is a slope, and the breaker is placed in front of the slope mouth, the main structure is

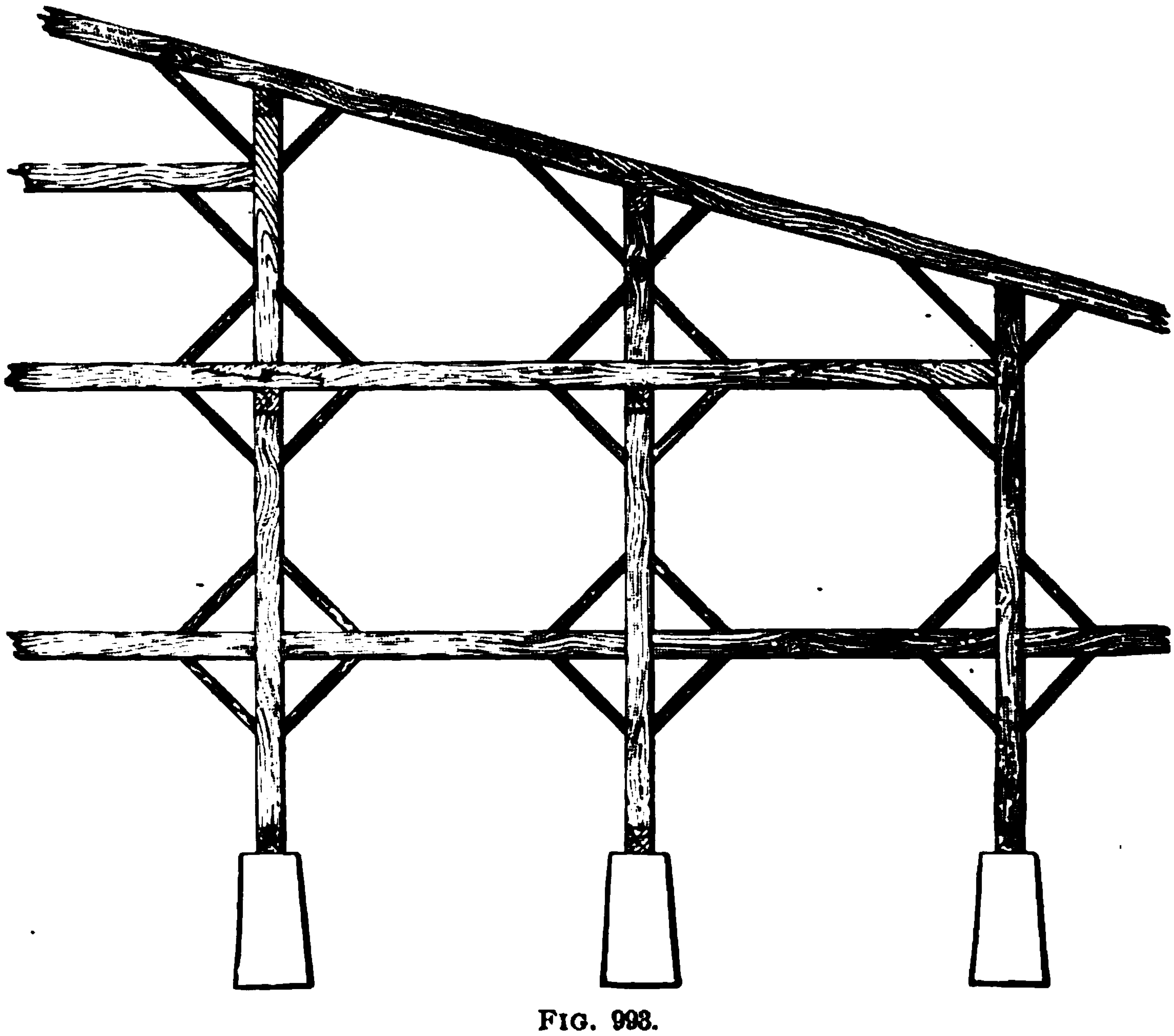


Fig. 998.

placed at some distance from the opening and is connected to it by an inclined plane, built as an open trestle and

forming a continuation of the slope. There are various methods used in framing these trestles, the particular form depending upon the height and the distance. Figs. 993,

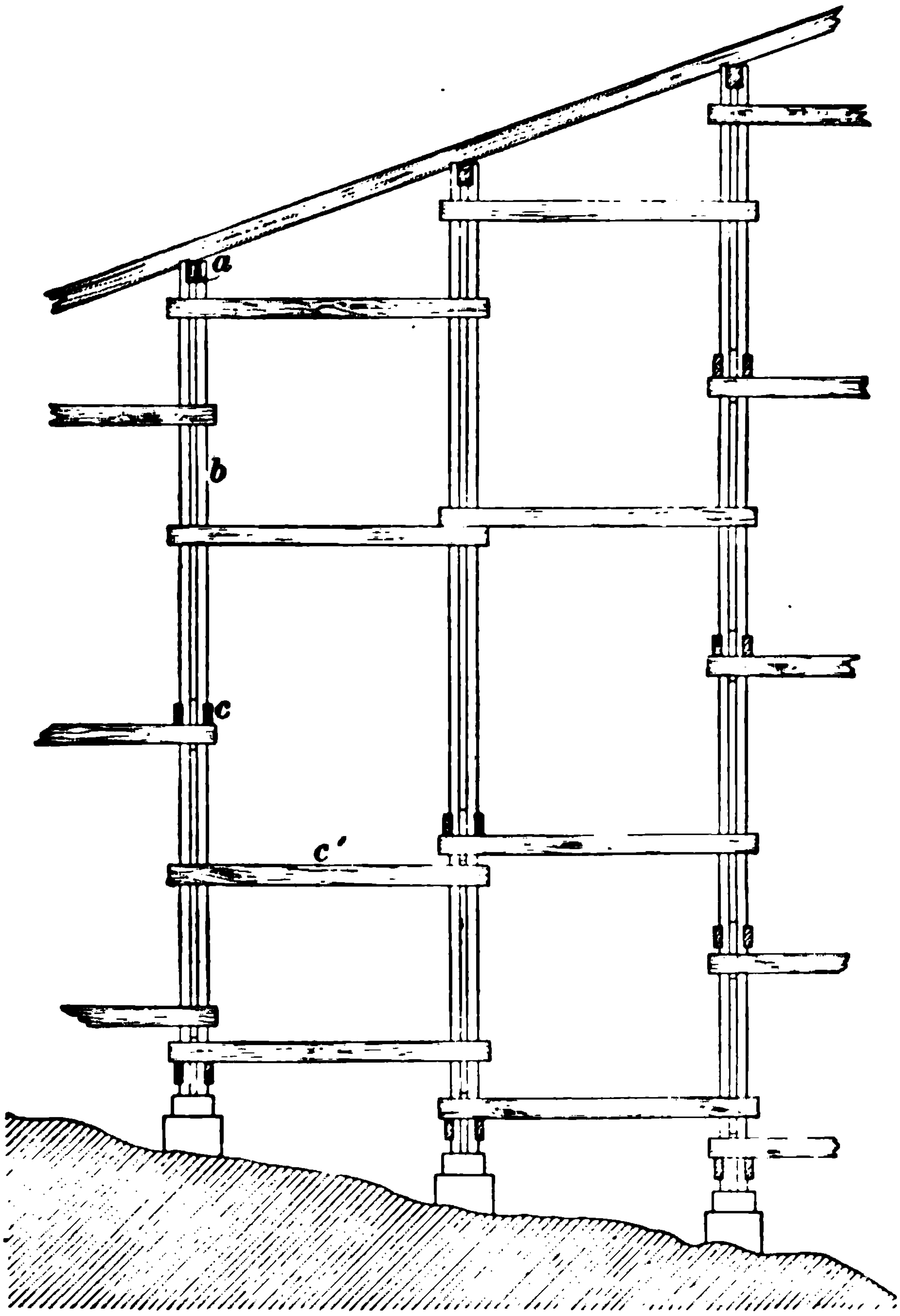


FIG. 994.

994, and 995 show the side elevation of some of the different forms for framing trestles used in building inclined planes in the anthracite region.

2762. The method of framing as shown in Fig. 993 requires very heavy timber, generally 12 inches × 14 inches

or 12 inches × 12 inches, with 5-inch × 6-inch braces. This method of framing makes a very substantial structure.

In Fig. 994 the timbering used is, *a*, 10 inches × 12 inches; *b*, 5 inches × 12 inches; *c* and *c'*, 5 inches × 10 inches. In this method the posts and the cross-beams are made up of two separate pieces of timber, the different parts being fastened together by bolts.

Fig. 995 shows a framing where **corbel blocks** ***a, a'*** are used. These give a greater bearing surface for the stringers, and, consequently, strengthen them. When the slope is

FIG. 995.

not on line with the breaker, there is what is called the **slope landing.** This consists of the tracks and turnouts laid on the ground at the mouth of the slope.

In some places, where it is necessary to get on higher ground to locate the turnouts, or where the cars can be run direct to the dump, some such arrangement as shown in Fig. 996 must be resorted to. This is simply a short inclined plane connected with a trestle.

2763. Safety Blocks.—Fig. 996 shows the arrangement of a safety block used at the head of slopes, in compliance with the Anthracite Mine Law. Such an arrangement of safety blocks is necessary at the head of every slope, to prevent the descent of mine-cars into the slope before the wire rope is attached. The amount of damage done by a single escaping car is often so great that the money required for repairs would pay for the adoption of an expensive

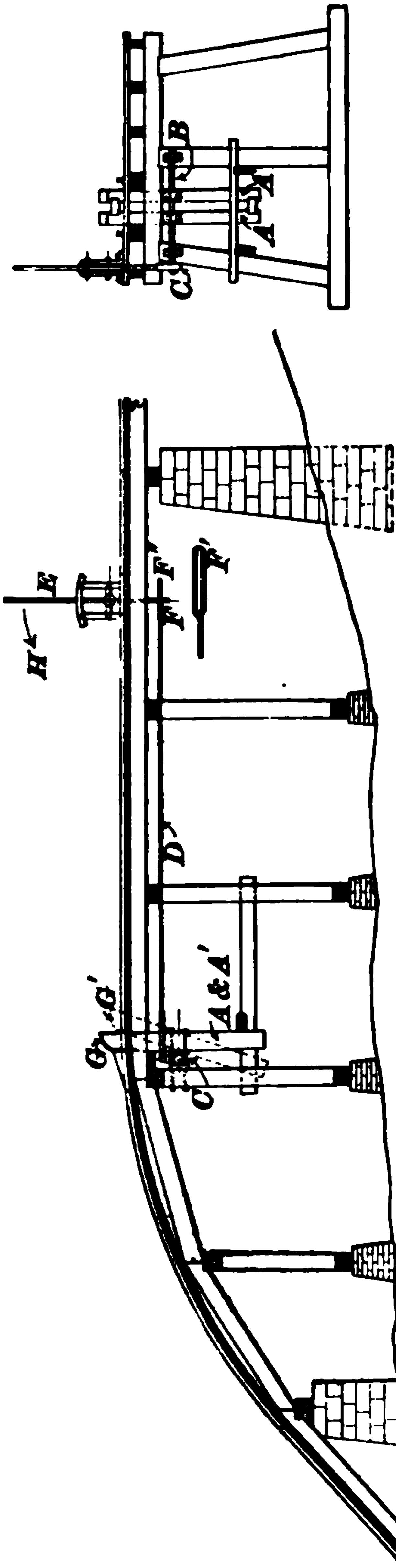


Fig. 996.

device for preventing such an accident. The block as arranged and shown in Fig. 996 is for a single-track slope. It is very simple, thoroughly reliable, and an inexpensive appliance.

This safety attachment consists essentially of the blocks *A* and *A'*, the shaft *B*, the arm *C*, the rod *D*, and the lever *E*.

The pieces *A* and *A'* are generally made of 8-inch × 12-inch timber, and are iron-bound at the extremities to prevent wearing. The shaft *B*, to which is keyed the arm *C*, is 2 inches to $2\frac{1}{2}$ inches in diameter, and the parts *A* and *A'* are securely fastened to it. *D* is a rod connecting the arm *C* with the lever *F*. The rod *D*, where it unites with the lever *E* at *F*, is made as shown at *F'*, in a sort of a loop. The car, as it comes over the knuckle, finds the blocks *A* and *A'* in the position *G*, and the rod *D* in the position *F*. The axle of the car strikes the block and changes the position from *G* to *G'*, and the rod *D* is changed from *F* to *F''*, on account of the loop shown at *F'*; hence, during this operation the lever *E* remains stationary.

After the car or cars have passed over the block, it resumes its vertical position, due to the

position of the shaft B. If it is found that the blocks do not resume their vertical position promptly, a weight can be attached.

When the cars are about to descend the slope, after the rope is attached, the man in charge of the lever E pushes it forward in direction H, thus bringing the block into position G'. This is a style of block that seldom gets out of order, and is simple and strong in its construction.

The above block is one of the many kinds in use in the anthracite region. The style generally depends upon the place where it is to be located.

FANS.

2764. There is no definite location on the surface for a fan in connection with the outside arrangements that helps to operate the plant. The location depends altogether on the arrangement of the underground openings.

Where a new plant is opened up by means of two slopes, which are termed the "main" and "tender" slopes, it is customary to place the fan between the two.

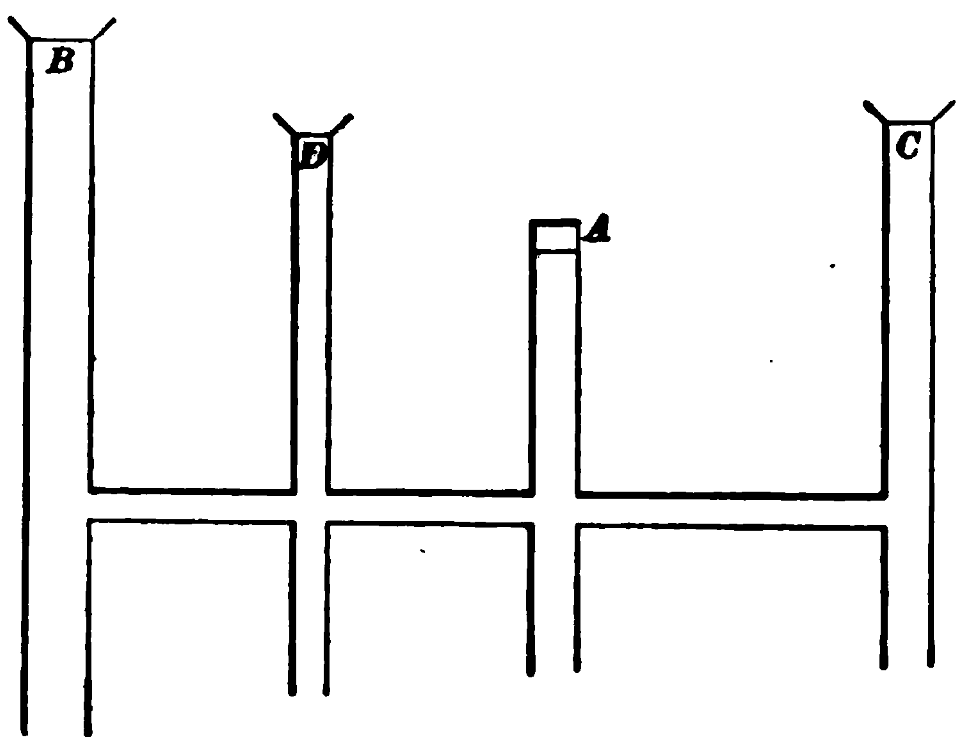


FIG. 997.

Fig. 997 shows the plan of opening up a plant by means of two slopes, the tender slope C and the main slope B. A is the shaft on which the fan operates, D being the pump way. In this method of opening, the tender slope C is driven downwards from the surface, while the openings A, D, and B are opened from below upwards.

In very deep shafts there is generally a fan compartment. In most cases, however, there is a special opening made

for the fan, either by sinking a shaft or driving a passage to the outcrop of the vein.

2765. Since the passage of the Anthracite Mine Law prohibiting the use of furnaces in gaseous mines, centrifugal fans have come into general use.

The centrifugal ventilating fan is a machine which is composed of a number of straight or curved vanes mounted on a shaft, to which a rotary motion is given. The air drawn from the mine enters the apparatus by an opening around the axis. It then comes in contact with the vanes, which communicate their motion to it, and under the action of the centrifugal force it is driven to the circumference and from there into the outer atmosphere.

Ventilating machines used at the anthracite mines act, in general, either as **exhausting machines,** placed at the top of the upcast shaft, or as **blowing-machines,** placed at the top of the downcast shaft. The exhausting machines draw in the air through a short or long channel and eject it into the atmosphere; blowing-machines, on the contrary, take the air from the atmosphere and force it into the mine openings.

2766. In the anthracite region there are various types of fans in use, but the one known as the Guibal fan is used more extensively than all others.

The most important and distinguishing feature of the Guibal fan is the spiral or circular housing. In this the fan differs from all others (except the Capell and the Schiele). The Guibal delivers all its air through one opening.

This opening is regulated by an adjustable shutter. The theory of the shutter is here briefly stated: The air is delivered at the throat at a velocity nearly equal to the ends of the vanes, and a certain known volume of air is delivered by the fan every minute. If the outlet is too small, the air does not find free exit; if too large, reentries of air occur behind each blade. The shutter provides a means of so regulating the size of the orifice that it is just sufficiently large to give free exit to the required quantity.

2767. The diameters of fans used in the anthracite region vary from 10 feet to 35 feet. From recent experiments, it has been found:

1. The only advantages obtained by increasing the diameter of fans are: Less speed is required from the engine if it works direct on the fan shaft, a proportional extra width is obtained, and there is a larger area for the air to flow through into the fan.

2. The width of fans appears to exert but small influence on their efficiency, but, as a rule, an increase in width enables a fan to exhaust more air.

3. The influence of the shape of the spiral casing is considerable. The best shape begins to curve away at or near the cut-off, and gradually increases the space between the blades and the casing until the outlet is reached. At this point it should be from one-fourth to one-third the diameter of the fan.

4. The influence of the shutter is decidedly advantageous, as by its use the opening at the end of the spiral casing is so regulated as to give the highest efficiency of the fan.

5. The tests show that fans give the best results when running at a peripheral speed of 5,000 to 6,000 feet per minute.

2768. In the anthracite region there are more different types of engines running fans than any other kind of machinery about the colliery. At the same colliery there often are horizontal, vertical, inclined, and oscillating engines running different fans.

Too much importance can not be placed upon the selection of a fan-engine for a large, gaseous colliery. A breakdown at any time may cause the loss of many lives, and result in great damage to the mines. It is, therefore, of great importance to have an engine that will run regularly and with little risk of breakage.

When the fan is located directly over the airway, it is subject to more danger from fire than when located a few feet away from the shaft mouth; but, even when fans are

not placed directly over the upcast, they are placed so near to it that the risk of damage by fire is almost as great.

To guard as much as possible against loss in case of fire, the fan enclosures should be built either of iron or brick, instead of wood.

ARRANGEMENT OF SOME OF THE BUILDINGS AND OTHER NECESSARY EQUIPMENTS.

2769. Engine-Houses.—Most of the engine-houses erected in the anthracite region are frame structures, with either shingle or corrugated-iron roofs; in many cases the sides also are of corrugated iron. At some collieries, where there are expensive winding-engines, the engine-houses are of brick or stone. Iron, stone, and brick are used in construction, to guard as much as possible against fire. The sides and roofs of frame and iron structures are coated with red mineral paint.

The different structures are all well lighted, heated, and ventilated, the ventilator being an opening in the roof fitted with small windows that can be opened. The engine-houses are generally heated by a few coils of steam-pipe, in connection with the steam fixtures that they always contain.

In constructing an engine-house, at least 5 feet should be allowed between the engine bed-plate and the sides of the building, and at least the same distance between the steam-cylinder and the end of the building. At the drum end of the building there should be a space between the end of the building and the drum fixtures, at least wide enough so that a man can cross from one side of the building to the other.

2770. Carpenter and Blacksmith Shops.—The carpenter and blacksmith shops should be located under one cover. This has always been found to be the better way in practice, for the one in many cases really depends upon the other. The building should be located in some place convenient to the main opening, so that it is not necessary to construct a great length of track to convey the cars from the shaft or slope to the shop for repairs.

The size of the structure depends upon the amount of work to be done. Very often larger shops are built at small plants than at some larger ones. This is because the mine-cars in the latter case are built at some other car shops, and all the heavy ironwork is also done away from the colliery blacksmith shop. When this is the case, the shops are erected merely for repair work, such as sharpening the miners' tools, repairing mine-cars, and any other work that may be needed about the colliery. At some collieries, where they build their own mine-cars and do their own heavy ironwork, very commodious structures are erected, which are fitted with the best class of machinery, so that the work can be turned out with neatness and despatch. Such shops are well lighted and ventilated, and the carpenter shop is built with a pit, so that men can work beneath the car.

In connection with the blacksmith and carpenter shops, some of the collieries have a well-equipped machine-shop in operation to do the necessary repair work about the colliery.

2771. Powder House.—This is the building wherein all the blasting material used about a colliery is stored. It is generally located in some out-of-the-way place (that is, separated from the other buildings), so as to avoid as much as possible the danger of explosion in case of fire; at the same time, it is so located as to be convenient for the miners.

The building is generally a frame structure, with a corrugated-iron roof. At some collieries in the anthracite region very substantial iron, brick, or stone structures are erected. The size of the structure depends upon the size of the plant. Very often a colliery is so located that it can be furnished with a new supply of explosives at very short notice, in which case a large stock need not be kept on hand.

Fig. 998 shows a powder house constructed of angle-iron and covered with No. 20 plain

Fig. 998.

painted iron. It is made in sections, so as to be portable, and can be fastened to any wood or stone floor, holes being made in the angle-iron for this purpose.

2772. Supply House.—This is the building wherein are stored the oil, cotton, shovels, and other articles used about the colliery. It is located, in some cases, very near the main opening, so as to be convenient for the miners; at other times it is located near the main railroad-track, for convenience in unloading barrels of oil when they are shipped direct to the colliery. This building is generally a frame structure.

There is another supply house at most collieries that is referred to as the **iron house,** which is located close to the blacksmith shop, as shown at *q*, Fig. 986. This building is a frame structure, usually 12 feet × 24 feet, and is used for storing bar iron, tool steel, bolts, etc.

2773. Office.—This is one of the necessary structures for a colliery. The office is generally divided into two compartments, one being occupied by the superintendent and the other by the colliery clerk and the shipper. At some collieries, where the engineering corps are stationed *at the mines*, they occupy a compartment in the superintendent's office. It is not customary at the present time for the colliery clerk to be stationed at the colliery. The larger mining companies generally have an office, located in a town or city near by, where the colliery accounts are made out and kept. The shipping-clerk, generally known as the **shipper,** then attends to everything pertaining to clerical duties about the colliery, and frequently he has an office all to himself. This office is usually a frame or brick structure located near the railroad shipping tracks, so that the shipper from his window in the office can keep a record of the loaded cars that are sent to market.

At many of the gaseous collieries there is a compartment in the superintendent's office in which there is a machine known as the **Shaw gas tester,** which is used to test the

air from the mine for the percentage of gas it contains. The air to be tested is brought out of the mine in rubber bags.

2774. Lamp House.—At many of the large gassy mines there is a frame building near the main opening known as the lamp house. Here the safety-lamps, are given to and received from workers in the mine by a man known as the **lamp man,** whose duty is to inspect, clean, repair, and keep a record of the different safety-lamps that are used in the mine. In mines where there is but a small quantity of gas the work of the lamp man is performed by the **fire boss** at some convenient place.

2775. Wash House.—This is a suitable structure upon the surface, as required by the Anthracite Mine Law, wherein the men employed in the mine can change their clothing before entering the mine, and can wash themselves and change their clothing upon returning therefrom. The structure is generally a frame one, and is located so as to be convenient to the principal entrance to the mine. The building is well lighted and heated, and is supplied with pure cold and warm water.

2776. Timber and Lumber Yards.—In the timber-yard the mine timber is sized preparatory to loading it into mine-cars or upon mine trucks. It is here, also, where the rails, sills, planks, boards, lagging, timber, etc., are stocked that are to be used in the mine. This yard is usually near the main opening, although at collieries where most of this material is received by rail, and there is a convenient place for unloading it, the yard is near the railroad-tracks. Then, there must be a track leading from the main opening to the timber-yard.

The loading track in a timber-yard should have a platform built on one or both sides of it high enough to bring the level of the platform a little above the height of the timber truck, so that there need be no unnecessary lifting

or rolling of the heavy timbers to get them upon the mine trucks. The lighter lumber to be used in the carpenter shop for building mine-cars or repair work is either cut at the colliery sawmill or received by rail. It is generally stored under small sheds near the carpenter shop, in a yard spoken of as the **lumber-yard.**

2777. Barn.—This structure is generally erected at some distance from the others, so as to be out of the way in case of fire. It is a two-story frame building. The first story, or ground floor, is used for stabling purposes, and contains the different stalls for the mules and horses used about the colliery. In the second story the large storage bins for the grain are located. The hay and straw used are also kept on this floor.

At some collieries most of the mules used underground are kept in underground stables, on account of the great inconvenience in bringing them to the surface. In such cases they are never brought to the surface until death overtakes them, or in case of a long suspension of mining operations.

At some collieries, where a large number of mules are used underground, the barn on the surface is made sufficiently large to accommodate them all in case of suspension. At other collieries the barn is made just large enough to accommodate the mules and horses that are used outside and those that have daily exit from the mine. In such cases, when there is a suspension of work underground, the stock is sent to some neighboring place for shelter.

The barn in Fig. 986 shows an ideal location. The second-story floor is just low enough to allow a truck to be used in conveying the grain from the car to the barn. A barn should be well lighted, and so constructed as to be at all times well drained. A dam should be built near the barn, to which the mules, as they come from the mines, can be driven and washed. If the mine dirt is allowed to accumulate on the mules, it will cripple them and, in time, make them unfit for service.

CULM, OR DIRT, OR WASTE AND ROCK BANKS.

2778. A suitable location for the culm, or dirt, or waste and rock banks is very often a difficult matter to decide upon. In determining the location of the rock and culm dumps, the topography of the surrounding area exercises a governing influence. It is always advisable to avoid depositing this refuse over the outcrop of a workable seam, for when these heaps catch fire there is more or less danger of the fire extending from the outcrop coal into the mine. It is also advisable to clear away all old logs, stumps, and other vegetable matter before dumping culm on a piece of land, as decaying vegetable matter in a culm pile will start a fire by "spontaneous combustion."

2779. At the present time waste may be divided into two classes: waste coming from *dry* breakers and waste coming from *wet* breakers. In the first case, the coal as it comes from the mines is in a more or less dry condition; in the second case, the coal as it comes from the mines is in a more or less wet and muddy condition.

In the first case there are separate rock, slate, and culm banks, or a combination of rock, slate, and culm; or, rock and slate bank, with a separate culm bank; or, slate and culm, with a separate rock bank.

In the second case, where the coal is wet and muddy as it comes from the mine, there are separate rock, slate, and culm, and slush banks; or, a combination of rock, slate, and culm, and slush; or, a combination of slush, slate, and culm, with a separate rock bank; or, a combination of slate and culm, with a separate rock and a separate slush bank; or, what is most general, a combination of rock, slate, and culm, and a separate slush bank.

2780. Previous to the last fifteen or eighteen years, the culm, or coal dirt, the slate, and bony coal picked from the coal in the breaker and the slate and rock coming from the mine were all deposited together in one heap; but the possible utilization of the culm or fine coal, either by burn-

ing in fire-boxes constructed for this purpose or by manufacturing into artificial fuel, has given to the culm as it lies in the banks a certain prospective value, and, to enhance this value and reduce the future cost of utilizing this refuse, two or three distinct kinds of refuse heaps are now common.

These old culm banks that are separated from the rock and slate banks, and which were deposited years ago when everything below the size of stove coal was called culm, are being used at the present day to operate what are termed **coal washeries.**

2781. In the first case referred to above, the coal during the separating and sizing in the breaker is kept dry.

The fine coal known as culm, coal dirt, waste, etc., that passes through the rice-screen mesh (or, when no rice coal is made, through the buckwheat-screen mesh, or when no buckwheat is made, through the pea-screen mesh), together with all the fine stuff made from the bars that are located directly under the main screens, is collected in a pocket or separate pockets and deposited in a heap known as the culm bank or dirt bank.

The rock and slate coming from the mine are carried directly to a heap known as the rock bank.

The slate picked from the coal in the breaker, after being collected in pockets in the breaker, is dumped on the slate bank, or, as stated before, is dumped with the rock or dirt.

The bony coal is sometimes collected in a pocket and dumped on the same bank as the slate, but generally it is crushed in an extra set of rolls called the **bony-coal rolls,** and made into the smaller sizes of coal.

2782. In the second case, the separating and sizing are performed by means of water. The coal is washed, and the culm coming from the rice-screen mesh, or buckwheat-screen mesh, or pea-screen mesh, as the case may be, is carried off by the water through troughs, and is deposited over an area known as the **slush bank.**

After the water leaves the trough, the heavier particles are

CULM, OR DIRT, OR WASTE AND ROCK BANKS.

2778. A suitable location for the culm, or dirt, or waste and rock banks is very often a difficult matter to decide upon. In determining the location of the rock and culm dumps, the topography of the surrounding area exercises a governing influence. It is always advisable to avoid depositing this refuse over the outcrop of a workable seam, for when these heaps catch fire there is more or less danger of the fire extending from the outcrop coal into the mine. It is also advisable to clear away all old logs, stumps, and other vegetable matter before dumping culm on a piece of land, as decaying vegetable matter in a culm pile will start a fire by "spontaneous combustion."

2779. At the present time waste may be divided into two classes: waste coming from *dry* breakers and waste coming from *wet* breakers. In the first case, the coal as it comes from the mines is in a more or less dry condition; in the second case, the coal as it comes from the mines is in a more or less wet and muddy condition.

In the first case there are separate rock, slate, and culm banks, or a combination of rock, slate, and culm; or, rock and slate bank, with a separate culm bank; or, slate and culm, with a separate rock bank.

In the second case, where the coal is wet and muddy as it comes from the mine, there are separate rock, slate, and culm, and slush banks; or, a combination of rock, slate, and culm, and slush; or, a combination of slush, slate, and culm, with a separate rock bank; or, a combination of slate and culm, with a separate rock and a separate slush bank; or, what is most general, a combination of rock, slate, and culm, and a separate slush bank.

2780. Previous to the last fifteen or eighteen years, the culm, or coal dirt, the slate, and bony coal picked from the coal in the breaker and the slate and rock coming from the mine were all deposited together in one heap; but the possible utilization of the culm or fine coal, either by burn-

ing in fire-boxes constructed for this purpose or by manufacturing into artificial fuel, has given to the culm as it lies in the banks a certain prospective value, and, to enhance this value and reduce the future cost of utilizing this refuse, two or three distinct kinds of refuse heaps are now common.

These old culm banks that are separated from the rock and slate banks, and which were deposited years ago when everything below the size of stove coal was called culm, are being used at the present day to operate what are termed **coal washeries.**

2781. In the first case referred to above, the coal during the separating and sizing in the breaker is kept dry.

The fine coal known as culm, coal dirt, waste, etc., that passes through the rice-screen mesh (or, when no rice coal is made, through the buckwheat-screen mesh, or when no buckwheat is made, through the pea-screen mesh), together with all the fine stuff made from the bars that are located directly under the main screens, is collected in a pocket or separate pockets and deposited in a heap known as the culm bank or dirt bank.

The rock and slate coming from the mine are carried directly to a heap known as the rock bank.

The slate picked from the coal in the breaker, after being collected in pockets in the breaker, is dumped on the slate bank, or, as stated before, is dumped with the rock or dirt.

The bony coal is sometimes collected in a pocket and dumped on the same bank as the slate, but generally it is crushed in an extra set of rolls called the **bony-coal rolls,** and made into the smaller sizes of coal.

2782. In the second case, the separating and sizing are performed by means of water. The coal is washed, and the culm coming from the rice-screen mesh, or buckwheat-screen mesh, or pea-screen mesh, as the case may be, is carried off by the water through troughs, and is deposited over an area known as the **slush bank.**

After the water leaves the trough, the heavier particles are

deposited first, and during the continuance of its flow it holds nothing but the finest sediment in suspension.

In order to take out as much sediment as possible before the water reaches the streams that drain the region, very cheap dams are constructed, called **brush dams.** These dams are made, as shown in Fig. 999, by piling logs, brush, etc., together and throwing earth back of them.

A number of troughs are used for overflows. The water accumulates in the dams, the sediment is deposited, and the water passes off comparatively free from sediment. At

FIG. 999.

some collieries no attention is paid to the removal of the sediment from the water; consequently, the streams are quickly clogged up, and a thick deposit covers the lower portions of the valley, giving rise to suits for damages.

As in the first case, the rock and slate that come from the mine are carried directly to a heap known as the rock bank.

The slate from the breaker and the fine culm that drops through the bars located directly under the main screens are conveyed to a pocket or pockets and dumped on the

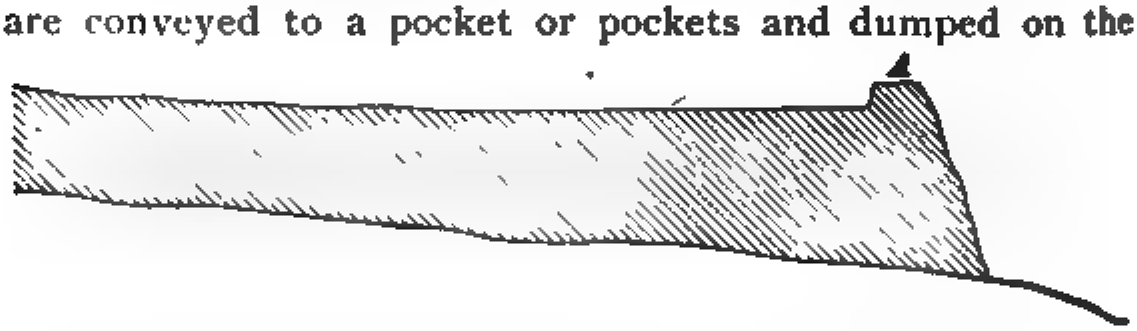


FIG. 1000.

slate bank; or, as before stated, the rock, slate, and culm are deposited on the same bank.

Fig. 1000 shows another arrangement of a slush bank, and how it is arranged to keep the deposit from getting into the stream. The slush as it comes from the trough is allowed to spread over a large area. Men are kept at work banking the deposit, as shown at *A*. The water either soaks through or is led off on the side of the deposit.

2783. At some collieries, where water is used in the breaker, there is not sufficient area on the surface for the slush banks already described. If no deposit is to reach any of the near-by streams, a machine is used, the end view of which is shown in Fig. 1001. It is simply a large tank, into

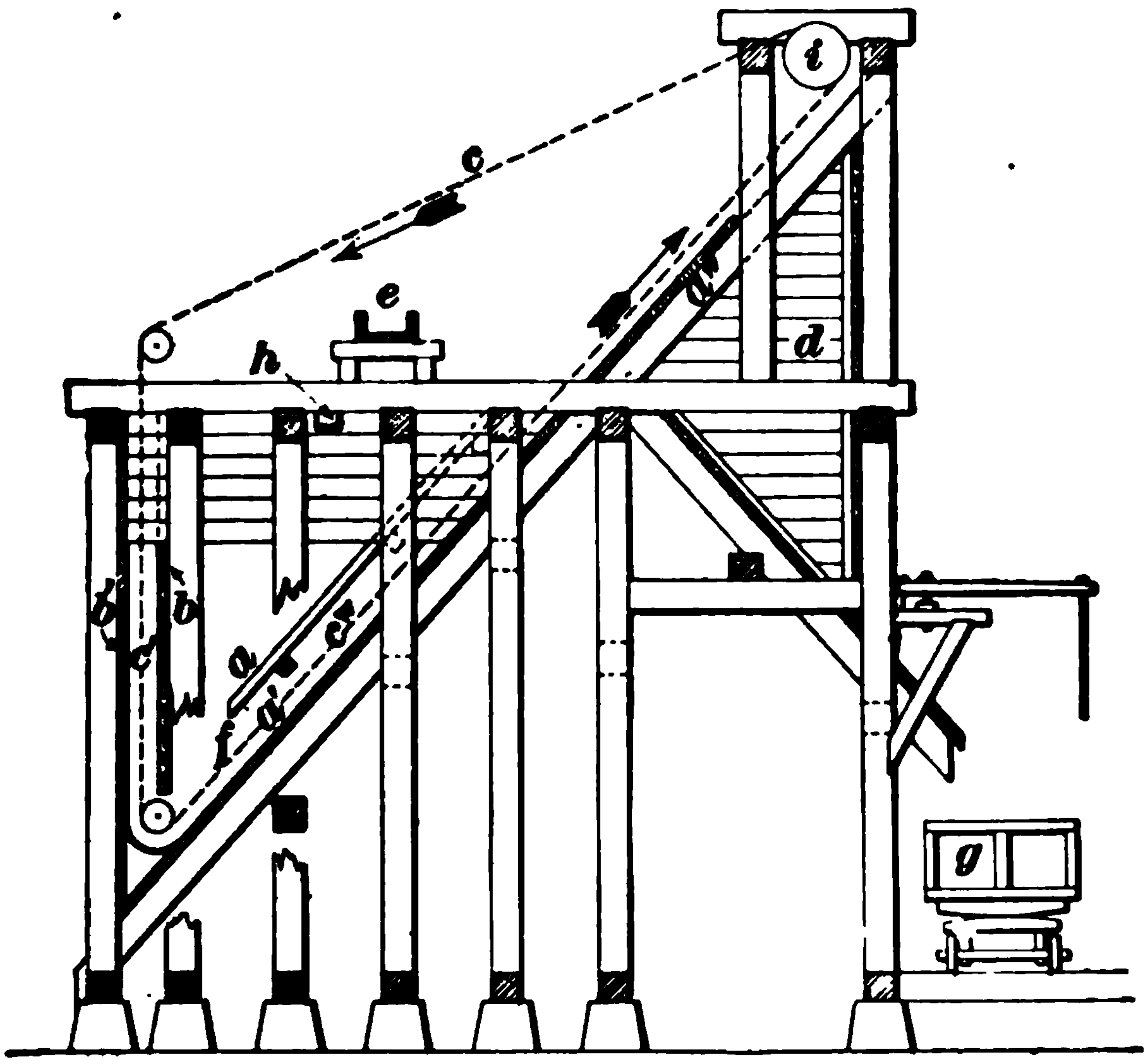


FIG. 1001.

which the water containing the culm is run as it comes from the breaker.

The tank is fitted with two bottoms, *a* and *a'*; *a* is termed the false bottom. It also has two sides, *b* and *b'*. *c*, *c'*, and *c''* is a line of drags, which are from 3 to 4 feet wide. In one tank there are from four to six of these drags, all working by means of shaft *i*; *d* is a pocket into which the culm is deposited as it is taken by the drags from the settling tank.

The operation of this may be explained thus: The water, as it comes from the breaker through the trough *e*, is deflected at intermediate points into the tank. The water in the tank is but very little agitated by the drags, which are kept going continually in the direction of the arrows. The agitation is but slight, because of the false bottom in the tank. The particles, as they settle, drop into the drag line at *f* and are conveyed to a'', where they drop into the pocket *d*, from which they are loaded into a dumper *g* and conveyed to the culm bank. The idea of extending the bottom a' to a'' is to drain off the water, so that it will not be carried into the pocket *d*. The water, as it leaves the tank through the overflow *h*, still contains the finest sediment. To remove this sediment from the water before allowing it to run into the streams, it is conveyed to large settling tanks, where it is allowed to settle. The very finest particles settle to the bottom, after which the water passes out through the overflow.

2784. Various methods are in use for conveying the culm and waste from the breaker to the culm and waste banks.

At slope collieries, opened on the edge of the basin, the ground generally falls away rapidly enough to gain ample dumping space within a short distance. In such cases the culm is collected in pockets located in the lower part of the breaker. Tracks are laid along the side-hill for a short distance from the breaker to a point where the dump is commenced. In almost every case, in such a location, it is found necessary to erect a short trestle to cross the empty and loaded railroad-tracks leading to the breaker.

By examining Fig. 986, it can be seen that in this location the ground falls away very rapidly, and a short trestle c'' is erected over the railroad-tracks.

At shaft collieries, and at slope openings on comparatively low ground, dumping height is obtained either by an inclined plane, usually known as a dirt plane, by erecting a system of conveyors, as shown in Fig. 1002, or by erecting a tower

Fig. 1003.

in connection with the breaker. In this last case the culm, or waste, is elevated to a considerable height and emptied into a large pocket that will hold from 18 to 20 tons. A trestle is built in connection with the tower, so as to make the culm pile some distance from the breaker.

In Fig. 1002 the conveyor shown is built to a considerable height, and discharges its product directly upon the heap. This system of conveyors is driven by the gearing at *a*. The waste is fed into the conveyor by a chute *b* coming from the breaker. The culm, or waste, as it comes out at the top, passes down the chute *c* and is "spread out" by the circular sheet iron *d*. In time it becomes blocked up, directly in front of the chute *c*. The culm is then led off on the sides of the heap by the circular sheet of iron shown at *d*.

At many of the collieries the line of conveyors, instead of discharging their product directly upon the heap, as just shown, discharge their product into a pocket, from which it is loaded into dump-cars and conveyed to the dump either by mules or by a small locomotive.

2785. At several of the collieries in the anthracite region where the above method is in use, the chute leading from the conveyor to the waste pocket has a set of bars which takes out all the fine stuff, and only the very coarse material reaches the pocket. The fine stuff that passes between the bars is conveyed by a chute to a bore-hole, through which it is washed into the mine, where it fills up the old workings and acts as an aid to the pillars that are left intact to support the surface.

At many of the collieries, especially where there are flat workings, large pieces of rock are never brought to the surface, but are stowed away in the abandoned workings underground. When this is done there are no rock banks on the surface, which gives an additional area that is frequently very desirable.

2786. Another system of removing culm from the breaker has of late years come into use where but very

little or no water is used in the preparation of the coal. This system is known as the **culm blower.** The arrangement is exceedingly simple and is a convenient one.

The waste is all collected in a pocket at the bottom of the breaker, and is fed through a hopper A, as shown in Fig.

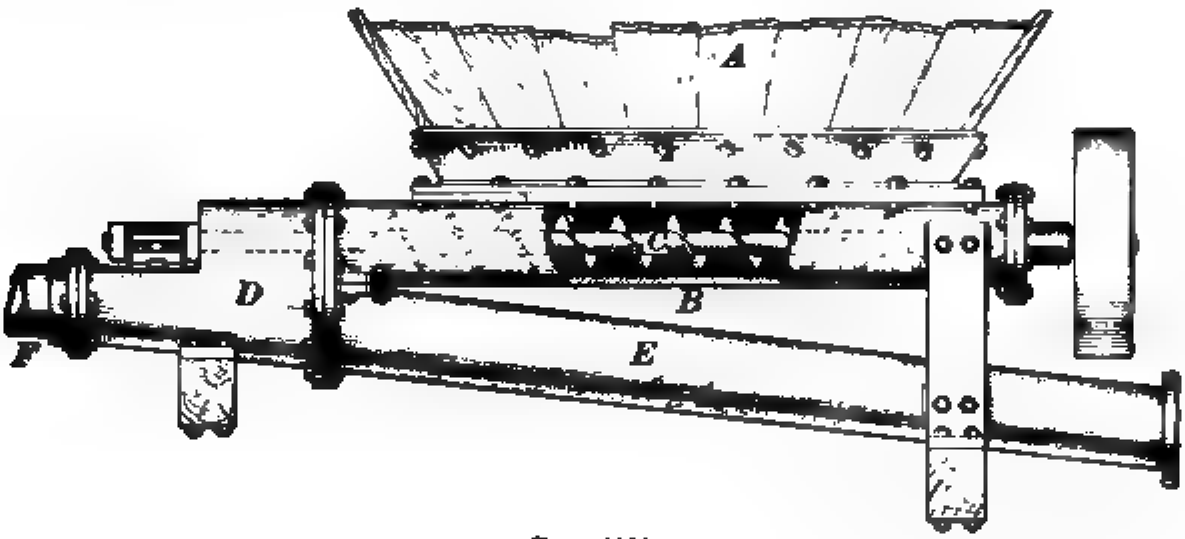


FIG. 1003.

1003, into a cast-iron casing B containing a worm C. This worm C feeds the waste into a chamber D, where it is met

FIG. 1004.

by a blast of air from the pipe E and carried through the column F to the culm heap, as shown in Fig. 1004.

2787. The pipe E is connected with an improved positive-pressure blower, whose inside arrangement is shown in

Fig. 1005. It consists of the two rotating bodies, or pistons, *a* and *b* keyed to parallel shafts *c* and *d*, which rotate in opposite directions with equal velocity. The form of the pistons is such that they touch each other like toothed wheels, and are enclosed in the casing *e*, which fits them as closely as practicable. From this figure, it is readily seen how the air between the pistons and casing is forced through the opening *f* into the air-pipe *E*, shown in Fig. 1003, while fresh air continually enters the casing through the suction opening *g*. In this machine *each* piston forces the air out of the space *V* twice during every revolution, so that the theoretical discharge per minute is $Q = 4\, n\, V$, where n represents the number of revolutions per minute of the machine,

FIG. 1005.

that is, of each of the two pistons. The piston shafts are driven by the belt-pulley *A*, and the cog gearing in the cases *B* and *C* cause the motion of one piston shaft to be transmitted to the other. These blowers are usually run at a speed of 200 to 250 revolutions per minute; in consequence of these high velocities, they are subjected to considerable vibration and work with a great deal of noise, to overcome which the cog gearing is run in oil, which is contained in the casings.

There is very little friction in the column *F*, Fig. 1004, and, consequently, comparatively no wear in the pipes, for the waste flows through the center of the orifice with a cushion of air all around it. The culm can be deposited at any place, simply by turning the mouth of the pipe in the direction in which the culm is to be deposited.

2788. The use of this blowing arrangement does away with the mules and small locomotives that are employed in all the other methods described, with the exception of the slush bank and the method of conveyors, as shown in Fig. 1002. At some collieries, where an inclined plane is used to do away with a number of mules or a locomotive, the dump-cars are hoisted up the plane, from which there is a descending grade to the dump.

To return the empty dump-cars to be loaded, a mule is used to pull them a short distance to a graded track cut in the side of the bank, leading to the foot of the plane or place where the dump-cars are to be reloaded.

2789. In locating a dirt plane, two cases have to be considered:

1. Whether it is better to use a **gunboat,** which is simply a large car that will contain two or three times the quantity that an ordinary dump-car contains; or,

2. Whether it is better to employ dump-cars on the plane where they are landed, either by using a barney or a chain with a hook or clevis attached.

Where a gunboat is used, a large pocket is built at the head of the plane, into which the gunboat empties. The gunboat is never detached from its rope upon arriving at the head of the plane.

2790. When the culm is handled as shown in Fig. 1006, which is a plan and elevation of a single dirt plane, a barney *M* is used. A barney is simply a small truck very solidly built, used to push the mine-car up an inclined plane or slope. In this case it pushes the dump-car *A* up the inclined plane *P*.

At the foot of this plane *P* is a small opening *N*, known as the **barney pit,** into which the barney *M* runs to allow the dump-car *A* to run over it in passing to the culm or waste pocket, the pocket for the culm being located at some distance from the foot of the plane.

This figure shows the arrangement and location of the sheave wheels at the head of the plane which lead the rope

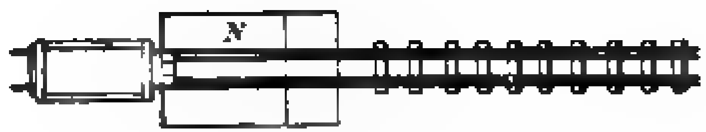


FIG. 1006.

to and from the haulage engine that is located at some distance from the culm heap, as shown.

The dump-car *A*, in coming down the inclined plane *P*, acquires enough momentum to carry it (after the barney *M* enters the barney pit *N*) up a slightly ascending grade on the empty track leading to the pocket, where it is again loaded with culm or waste. In returning, it comes in contact with a spring switch that transfers it to the loaded track, which is on a descending grade to the foot of plane *P*. On the arrival of the dump-car at the foot of the plane, the barney comes out of the pit and pushes the dump-car up the plane. The dump-car, being free from the hoisting rope, is landed very nicely and allowed to run to the loaded turnout, from which it is conveyed to the dump either by mules or a small locomotive.

2791. On a plane where the dump-cars are hoisted by using either a hook or clevis attachment for fastening the rope to it, the chain at the foot of the plane either remains attached to or is detached from the dump-car.

In case of either gunboat or dump-car, the foot of the plane should, if possible, be so arranged that the gunboat or dump-car can be run directly to the pocket for loading, without detaching.

The plane should also be so located, if possible, as not to face the main structure or any other important building that might be injured by the gunboat or dump-car breaking loose from the rope.

The engine for a dirt plane is sometimes located on the waste heap directly under the head of the plane, the bedplate of the engine being set upon a timber frame resting upon a timber cribbing. This is very poor practice, for in many cases the bank on which the engine is located takes fire, causing a continual settling of the crib, and making it impossible to keep the engine in proper running order.

It is always better for the engine to have some such position as shown in the figure, where it can have a good solid foundation, even if it does require an extra length of rope and some extra sheave wheels.

2792. The waste heaps often become of such dimensions as to encroach upon the immediate surroundings, making a very perplexing problem to overcome.

When the site is first selected for the waste, it should be in a location that will not interfere with the enlargement of the colliery plant, in so far as the erection of buildings is concerned; hence, buildings connected with a colliery should not be located so as to interfere with the growth of the waste banks.

At many collieries there is an ideal location for waste heaps, but at the same time this location may contain the outcrops of all the workable seams. It is better to sacrifice such a location and adopt a more expensive one for the waste heaps than to run the risk of destroying the whole mine by fire from the culm pile.

The advisability of having separate waste heaps presents itself in case of long suspensions of work, for the culm can then be conveyed directly to the boiler house from the culm bank without any previous preparation.

TRACKS.

2793. Railroad-Tracks.—One of the important matters that must receive attention at a colliery plant is the location of the empty and loaded turnouts for the cars that are used to ship the coal to market. This is often a very serious matter where the topography of the surface is such that, unless large sums of money are expended in excavating, very little room for tracks is obtainable.

Such a case is shown in Fig. 986, where the topography rendered it necessary to make a very extensive cut between the breaker and the main-slope hoisting-engine, in order to get the main line connected with the empty turnouts.

The tracks should be so arranged that very little shifting is necessary.

Assuming that the tracks are kept in a good condition, the grades should be such that a car will move readily by gravity as soon as the brake has been loosened, without the aid of barring.

To accomplish this, the empty track should have a grade of at least 2 feet in 100 feet, lessening to a grade of 1.5 feet at the entrance to the breaker.

The grade for the loaded tracks can be somewhat less; 1.25 feet in 100 feet is sufficient to run loaded cars over.

In every case the railroad facilities should be such that the receipt of the empty cars and the despatch of the loaded cars can be accomplished with the least possible difficulty.

2794. Mine-Car Tracks.—The general construction of mine-car tracks for the surface arrangement of mines does not differ to any marked extent from that of a large railroad, except that less attention is paid to the detail work; besides, the parts that go to make up the track are of very much lighter material.

In the anthracite region there is no standard gauge for the mine-car tracks. The gauges most commonly in use are 2 feet 6 inches, 2 feet 9 inches, 3 feet, 3 feet 6 inches, 3 feet 9 inches, and 4 feet. Intermediate gauges have also been used.

The grades of the tracks for the empty and loaded cars to run upon by gravity depend, in many cases, upon the mine-car, for there are some very easy and some very hard running cars. For an easy-running car to run by gravity, the empty track should vary in grade from 2 to 1.25 feet in 100 feet, and for loaded cars the loaded track should vary in grade from 1.25 feet to 0.75 foot in 100 feet.

The radii for the curves should be as large as possible, and never less than 25 feet.

The frogs used in connection with the track are usually made from rails. The tongue of the frog is made of two short pieces of rail, cut and riveted together so as to form the required frog angle. The wing-rails are a part of the switch rails leading away from the frog, but are bent to suit the frog angle by means of a rail-bending machine.

To avoid putting in a frog, a **cross latch** can be used. This is a short piece of rail with an eye in one end, so arranged that when it is put in place it can be thrown

across the one rail of the track that the car is to be put on or taken off.

The switches in many cases are the ordinary movable rail switch, but the one most commonly used is known as the **latch,** or **tongue,** switch.

The latches are wedge-shaped bars of iron with an eye in the thick end. The point and eye end of the tongue are set on small iron plates which are fastened to the cross-ties.

The latches are sometimes connected by a rod, so that they can be opened or closed at the same time. This switch in many cases is made self-closing, or automatic, by attaching the latches by a bar or lever to a metallic spring, an elastic stick of wood, or to a counterweight.

The cross-ties used vary in dimensions according to the gauge and the amount of traffic. Where a small locomotive runs over the track, the road is laid with good wide cross-ties. Ordinarily, hewed ties from 4 to 6 inches thick and from 5 to 8 or 9 inches wide are used.

The rails used are the regular **T** rails, varying in weight from 20 pounds to 50 pounds per yard.

The fish-plates are often the same as those used on a regular steam road, but in many cases they are made by the colliery blacksmith from old scrap iron that is gathered about the mine.

WATER-SUPPLY.

2795. The establishment of an adequate water-supply is often a difficult matter at anthracite collieries, and one that entails a great deal of expense. The water-supply at many of the collieries is obtained by damming up the small mountain streams. When this is done, the dam is usually above the level of the colliery, to obtain a sufficient head for forcing the water through the pipes to the different places where water is required about the plant; but in districts depending upon small mountain streams there is always a scarcity of water in dry seasons, and sometimes during very cold weather.

The larger streams that drain the coal fields are usually

so contaminated with the water that is pumped from the mines and with that coming from the coal washings that it is absolutely useless for steam purposes.

When these streams are fit for use dams are constructed, which, however, are generally on the same level or below the level of the colliery plant, so that a pump is needed to raise the water to the required height. At some of the collieries the water is obtained from a large stream located outside of the coal-producing area; at others it is obtained by sinking artesian wells. Of late years, many of the collieries purchase their water from some established water company, which in many cases pipe it a very long distance. As a last resort, the mine water is purified. The amount of sulphuric acid in mine water varies considerably. At some mines it has been known to reach 100 and even 200 grains per gallon of water. Such water will destroy iron with alarming rapidity, and must not be used in boilers under any circumstances. Water containing only two or three grains to the gallon has been known to ruin boilers in a few months.

2796. Where the water is purified, it is found that lime is the cheapest and best alkali to use, because the sulphate formed is least soluble. Soda or potash will serve the same purpose as lime, but the sulphates formed are entirely soluble in water and produce large deposits on evaporation. The sulphate of lime is also soluble, but to a very slight extent, 1 part of sulphate of lime being soluble in 400 parts of water. The reaction which takes place when lime is added to mine water can be expressed by the formula:

$$\underset{\text{Sulphuric Acid.}}{H_2SO_4} + \underset{\text{Burnt Lime.}}{CaO} = \underset{\substack{\text{Sulphate of Lime,}\\ \text{Precipitated.}}}{CaSO_4} + \underset{\text{Water.}}{H_2O}.$$

The process may be described as follows: The mine water is pumped into a tank or a series of tanks, and lime, slacked and reduced to a creamy consistency with water, is added, the amount varying from $\frac{1}{4}$ of a peck to $\frac{1}{2}$ of a bushel of lime for every 4,000 gallons. When sufficient lime has been added,

the contents of the tank are stirred until thoroughly mixed, and the mixture allowed to settle until perfectly clear. The time required for settling varies from ½ hour to 5 hours, according to the amount and the nature of the deposit. After settling, the water is drawn off into another tank or a series of tanks, from which it is pumped or injected into the boilers.

The deposit, or settling, is then removed from the tank, which is refilled and again operated upon.

Care should be taken to add just sufficient lime, so that the water will have no effect on litmus paper, either the red or the blue. If insufficient lime has been added, the blue paper will turn red, and if too much, the red will turn blue. Litmus paper is the best test to use for acid or alkali in this process.

2797. A positive test for detecting sulphuric acid is as follows: Add to the water suspected of containing sulphuric acid a strong solution of chloride of barium ($Ba\,Cl_2$); then, if any acid is present, it will be precipitated in the shape of sulphate of barium, a white powder; but, for purifying mine water upon a large scale, blue litmus paper is a sufficiently delicate test for acid. The paper is put into the water and moved around for half a minute. If the color of the paper does not change to red, no acid is present. This is not a positive test when minute quantities of acid are present; however, when such water is evaporated in a boiler, and blue litmus paper is placed at the open gauge-cock, it will turn red.

2798. As previously stated, the quantity of lime necessary to be added is gauged with litmus paper, the change of the blue paper to red showing that too little lime has been used, and the change of the red to blue showing that too much has been employed. It is in carefully guarding against the use of too little or too much lime, and thus obtaining water for use in the boilers as nearly pure as possible, that the secret of success lies. It is better, however,

that too much rather than too little lime should be used. If there is not sufficient lime, the acid will eat away the iron, which can not be replaced; but if there is a small excess of lime, the only consequence is a little more dirt in the boilers. In making steam from water purified by this process, a small deposit will form in boilers, which will require them to be cleaned out once a month. If, however, the water is put through a heater before entering the boilers, and raised to a temperature of from 280° F. to 320° F., it will be rendered as fresh and pure as rain-water, and will neither eat the iron nor form scale or mud.

THE PREPARATION OF COAL.

INTRODUCTORY.

2799. Anthracite coal, as it comes from the mines, is not marketable. "The run of the mine" can not, as in the case of bituminous coal, be sold. As it comes from the mine, the coal consists of (1) fragments of all sizes, mixed with more or less slate and rock; (2) more or less coal known as **bony coal** (slaty or argillaceous coal), and (3) lumps of coal with layers of slate adhering to one or both sides, or distributed throughout the lump, commonly known at the mines as **chippers.**

2800. To place the coal upon the market for domestic and manufacturing purposes, so that it will meet the approval of the consumers, it is necessary to subject the coal to a more or less complicated process of preparation, which has for its principal objects:

1. The removal of rock, slate, chippers, bony coal, and other impurities which are present in the coal as it comes from the mine.

2. The assortment of the coal into grades of nearly uniform size.

3. As there is a larger demand for coal of the intermediate sizes than can be supplied from the coal as mined,

it is necessary to break up some (or all) of the large lumps, so as to increase the percentage of the intermediate sizes.

All these objects are accomplished by the process of preparation in the breaker, which, according to the definition already given, is the structure containing the machinery used for the preparation of coal.

The purpose, then, of a breaker is to remove the impurities as completely as possible, and separate into the different sizes the coal that is to be put upon the market. This can be done either (1) by hand labor or (2) by mechanical means.

2801. In the first case, the coal is passed along chutes, on the sides of which men and boys are seated, who pick out the slate and, in some cases, the bony coal and the chippers. The bony coal and the chippers are separated only from the larger sizes of coal, varying from lump coal to stove coal, and are reprepared by breaking into sizes below stove coal.

In bony coal, the coal and slate are so interstratified as to destroy or greatly diminish its market value; while in the case of chippers, the coal and slate are so arranged as to give very little trouble in obtaining marketable sizes.

The chippers from the larger sizes (lump and steamboat) are prepared by what is termed **chipping.** Here the slate is separated from the coal by manual labor simply by using a pick, or in many cases by using a specially designed tool called the **pick hammer.** Below the sizes of lump and steamboat, the chippers and bony coal are generally prepared by the same set of rolls, known as the **bony-coal rolls.**

2802. In the second case, where the slate is separated from the coal by mechanical means, the operation will depend upon one of three physical characteristics of the coal and slate: (1) The difference in their specific gravity; (2) the difference of the forms into which they break, and (3) the difference of their angle of friction, or, in other words, the difference in the angle of a chute, lined with stone or iron, down which the coal or slate will slide without any increase of velocity. As a rule, slate will not slide down a chute which will carry coal.

The first of these refers to the jigging of the coal, and this operation may be performed by using either a wet or a dry jig. The wet jig is in use almost exclusively in the anthracite region, and will be described later.

The second characteristic refers to the difference of form into which they break; coal in breaking assumes a more or less rounded form, while the slate is broken into flat pieces. By running these two different forms over a chute, in which bars with longitudinal openings are placed, most of the flat pieces of slate will drop out through the small openings and the coal continue to the pocket or place of storage.

The above operation is also performed by a cylindrical screen, known in the breaker as the **slate-picker screen.**

The third characteristic: since slate will not slide down certain chutes that will carry coal, small openings are made in the bottom of the chutes at intervals. The coal glides over these openings, while the slate, which moves with a different velocity than the coal, in coming down the chutes at a higher velocity than the slate, simply dragging sluggishly along, drops through. In many cases sandstone slabs are used, which are placed in front of the openings in the bottom of the chutes.

2803. The method of preparation of coal is not the same throughout the anthracite coal fields of Pennsylvania. This is due to the different ways in which the coal-beds are deposited.

In the Wyoming and Lackawanna regions of Pennsylvania the coal seams lie more or less horizontally, and the coal, after it has been mined, is more or less prepared by the miner or the laborer while it is being loaded into the mine-car.

The miner or his laborer, in loading, is obliged to handle all the coal by hand; that is, he throws the large lumps into the mine-car by hand, and uses a shovel to get the smaller lumps into the car. While doing this, the larger pieces of rock and slate are thrown to one side to be stowed away in the underground openings of the mine, or they are placed

upon a heap, and afterwards loaded into a car to be hoisted to the surface and there deposited on the rock dump.

In the Lehigh and Schuylkill regions, where the beds have a more or less steep dip, the coal, after being mined, is loaded into the mine-cars from a chute or platform. The coal-beds in these regions are frequently wet, as compared with the coal-beds of the Wyoming region; consequently, the coal as loaded into the mine-cars consists of coal, rock, and slate, the whole being covered with a black muddy mass of fine coal and dirt, dripping with dirty water, and looking as it comes from the mine like anything but marketable fuel.

From the above, it can be readily seen that different methods of preparation must be used. On the one hand the coal is prepared dry, on the other it is prepared wet. The latter means that the coal as it passes through the different operations of preparation is treated with water to wash off the mud that adheres to the coal.

When water is used in the course of preparation, the method is more expensive than where the treatment is dry throughout; for the former requires extra machinery, and, since the water used is in many cases pumped out of the mine, the acid in it soon destroys the machinery and other parts with which it comes in contact.

THE ANTHRACITE COAL-BREAKER.

2804. The evolution of the anthracite coal-breaker forms an extremely interesting chapter in the history of the Pennsylvania coal trade.

In the early history of the côal trade, the coal was shipped to market as it came from the mine, the consumer breaking it up with hammers and screening it himself; but the loss of the fine coal, which was practically useless, since it could not be burned in an ordinary grate nor in any appliance suited to the combustion of coarse coal, was so heavy, and the trouble, annoyance, and cost of breaking up the coal by hand became so great, that it led to the introduction of machinery to break and screen the coal and classify it according to size.

Innumerable devices have been experimented with for breaking the coal, commencing with the hammer, passing through various types of toothed plates, to rolls with teeth, and corrugations of various forms.

2805. As the machinery has been improved the waste in preparation has been diminished, not only by better and more careful handling, but also by utilizing a larger proportion of the coal. It is not very long since *chestnut* was the smallest size of salable coal, while everything below that size went to the culm pile. The introduction of improved forms of grates and furnaces has added to the market list the following additional sizes: *pea*, *buckwheat*, and *rice*, which find extensive use in the production of steam, thus enforcing economy both in production and consumption.

2806. Anthracite breakers, in some cases, occupy as much as 23,000 square feet of surface, and vary in height from 50 to 145 feet, according to the method in use for the preparation of the coal. It is always desirable to handle the coal by gravity, allowing it to slide down chutes from each set of bars (or rolls) or screens, until it reaches the pockets. Hence, it is always better to have the necessary height, instead of elevating the coal a second or a third time by a system of elevators.

In capacity, anthracite breakers range as high as 4,000 tons per day. By this is meant the coal loaded into railroad-cars after it has been prepared in the breaker.

The capacity in every case depends upon the screening surface and the rapidity with which the coal can be run through the breaker, for it is generally found in practice that the rolls are capable of crushing more coal than the screens can handle.

2807. The general plan of a breaker structure is in the form of a wooden trestle, although of late years iron breakers have been introduced. Where iron is used, the breaker is a pin-connected structure, the posts being of cast iron, the struts generally of cast iron, and the rods of wrought

iron. Most of the large beams are riveted-plate girders; the smaller ones are rolled.

These iron structures are generally built for the purpose of guarding against destruction in case of fire. The framing of a breaker structure must be of the most substantial character; the heavy machinery that is required, the weight of the coal undergoing preparation, and the large surface presented to the force of the elements show the necessity for strength.

The timber ordinarily used in their construction is yellow pine, white pine, hemlock, oak, and birch.

The posts are generally of white pine or hemlock, and are usually of the following dimensions, in inches: 12 × 12, 12 × 14, 14 × 14, 14 × 16, 8 × 8, and 6 × 8. In some of the breakers recently constructed the posts are double; that is, they are made up of two pieces of timber, which can be renewed by taking out one side at a time, as shown in Fig. 994. Where these posts are used, they are usually of the following dimensions: 10 × 12, 5 × 12, and 5 × 10.

The sills are usually of oak, and in dimensions to suit the above sizes of posts.

The braces are of hemlock and white pine, and are usually of the following dimensions: 4 × 6, 6 × 8, and 6 × 12.

The stringers are usually either of white and yellow pine or hemlock, and of different dimensions. Birch is used in the form of boards for lining the bottoms of the coal pockets.

Further on is shown the plan, elevation, and cross-section of an anthracite breaker; but before entering into a description of it, the different methods of getting the coal into the breaker and the machinery used in preparing the coal will be discussed.

METHODS OF GETTING THE COAL INTO THE BREAKER.

2808. The coal as it comes from the mines is almost invariably conveyed to the top of the breaker, where it is dumped into a chute known as the **dump chute,** which contains a set of inclined bars, over which the coal runs; or

it is first dumped into a chute or pocket, from which it is slowly fed under a gate and allowed to slide down over the bars.

In case the opening is a shaft and the breaker is located the required 200 feet from the shaft, one of the methods of getting the coal to the top of the breaker is by means of what is termed a **vertical hoist**, Fig. 1007. This figure shows the plan and elevation of a double vertical hoist. The coal after reaching the surface is conducted from the mouth of the shaft to the foot of the vertical hoist, either by gravity or by some one of the many other methods of transfer.

The vertical hoist, or hoistway, contains two self-dumping cages. These cages are built of angle-iron and are riveted throughout. The

cage is made up of two parts, *a* and *b*, united by the hinge *c*. The part, or platform, *d* which carries the car is fastened to the part *a* by the support *e*, and is further secured by the braces *f*. To the part *b* the rope is securely attached. To the part *a* is attached a small wheel *g* which runs along the rail *h*. This wheel *g* keeps the parts *a* and *b* of the cage closed when the cage is in a vertical position, as shown at the foot of the hoist. At the top of the hoist the rail *h* is bent, as shown, and as the wheel *g* follows the rail *h* up the hoist, at the point *i* it is made to follow the rail *h* by means of the guard or deflection rail *j*. As the wheel *g* follows the curve of the rail *h*, the parts *a* and *b* become separated, and in so doing the self-dumping part of the cage comes into play. The part *b* of the cage is kept in a vertical position by means of shoes which act on the cage-guides *k*.

2809. The plan shows the arrangement of the tracks and guides. The car as it comes from the mine runs on the cage, as shown, from track *l*. As soon as the cage is raised from its position in the pit, the wheels of the car drop into the opening *m* made by the rails *n* shown in plan; this fastens the car to the cage, so that there is no danger of it leaving the cage when it is being dumped.

The car, while being dumped, can be given different angles of inclination, simply by notifying the engineer in charge of the hoisting-engine to raise or lower the part *b*. As *b* is raised or lowered, the part *a*, by means of the wheel *g*, assumes different positions.

The vertical hoist varies in height from 50 to 145 feet, according to the height of the breaker, and can also be used when the main opening is a slope, drift, or tunnel.

In the operation of dumping, only one man or boy is required. He attends to the opening of the latches on the car, also to taking off the tickets which are used to designate by whom the coal was mined.

2810. A plan of one of the many different methods of transferring the cars from the shaft opening to the vertical hoist is shown in Fig. 1008. The coal that is handled by the

breaker *A* is elevated to the top of the breaker through the vertical hoist *B*. The coal handled is mined through the two shafts *C* and *D*. The loaded tracks *E* and *F* leading away from these shafts are on a descending grade from the shaft mouth to the foot of the vertical hoist. The

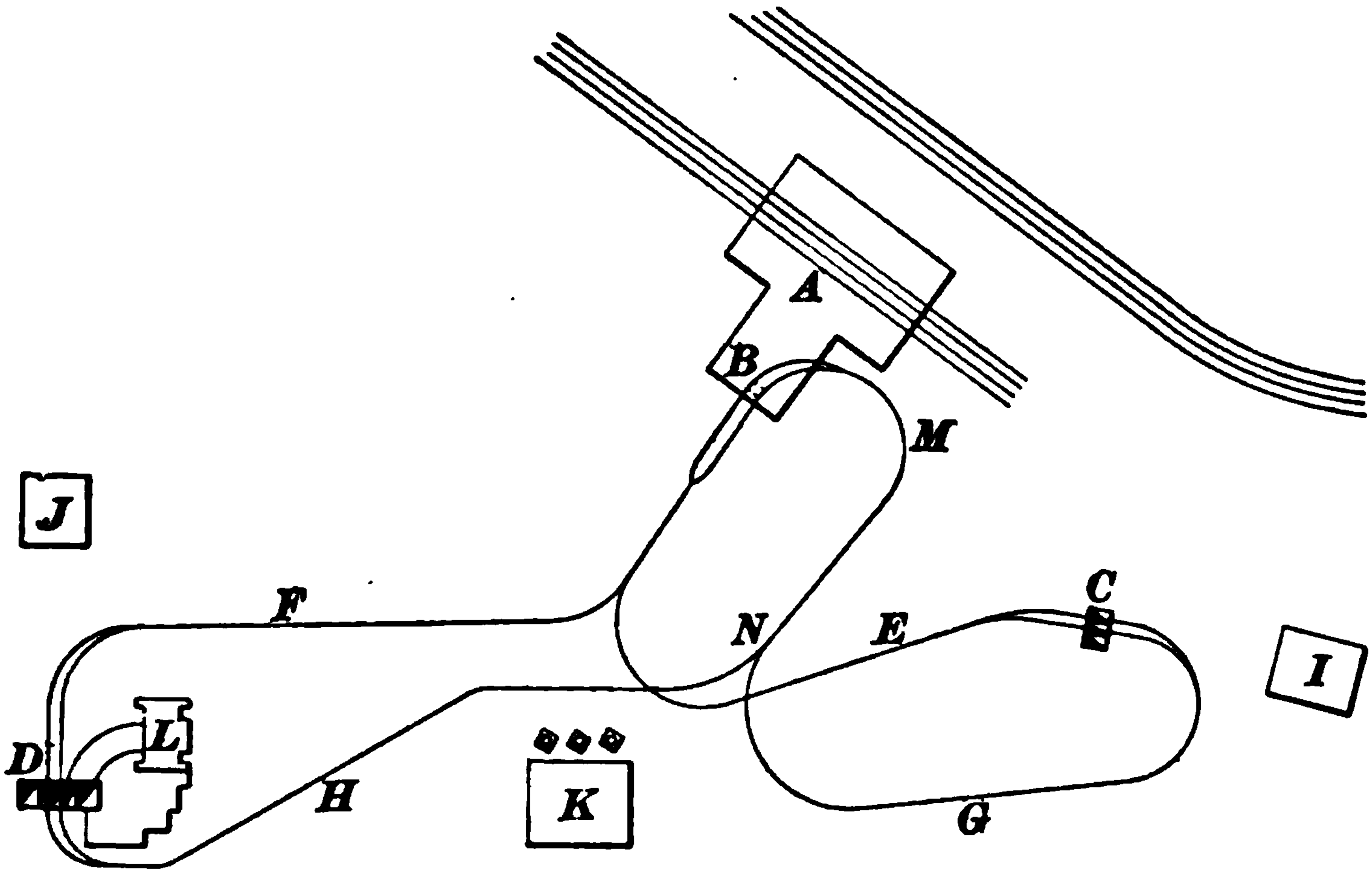


FIG. 1008.

loaded track *E* has a descending grade of $1\frac{3}{4}\%$, while the loaded track *F* has a descending grade of $1\frac{6}{10}\%$.

The empty cars, after leaving the vertical hoist, are run to the foot of the inclined chain hoist *M N*; at the top of this chain hoist are the two empty tracks *G* and *H* leading back to the mouths of the shafts *C* and *D*. Both the empty tracks are on a descending grade of $1\frac{3}{4}\%$ from the top *N* of the chain hoist to the mouths of the shafts *C* and *D*.

In this plan, *I* is the engine-house containing the winding-engine for shaft *C*, while *J* is the engine-house containing the winding-engine for shaft *D*. The location of the boilers that are used to furnish steam for these engines is shown at *K*, while *L* shows the location of the ventilating fan.

2811. Another method of getting the coal to the top of the breaker from a shaft opening is shown in Fig. 1009, where an iron or steel chute is built in connection with an iron or steel head-frame. The hoistways in connection with

this head-frame are fitted with self-dumping cages, the car never leaving the cage while it is being dumped. The coal is then elevated direct from the underground workings through the shaft opening and dumped into the chute which conveys the coal to the top of the breaker.

In order to comply with the Anthracite Mine Law, the breaker is located at a distance of 200 feet from the shaft opening, as shown. The height of the breaker, together with the extra height that must be allowed for the inclination of the chute in a distance of 200 feet and over, necessitates the building of a very high head-frame.

In Fig. 1009 the head-frame is 52 feet square at the base and 187 feet high to the summit. At a height of 149 feet the chute is reached, and at this point the cages stop. The length of the chute is 216 feet. It has a capacity of 100 tons

Fig. 1009.

of coal, and is generally kept nearly full to prevent excessive breakage of the coal. Besides being supported at one end by the breaker and at the other by the shaft head-frame, or tower, it has two steel towers as additional supports.

2812. Fig 1010 shows another method of getting the coal into the breaker where the top of the breaker is below the level of the opening, thus allowing a descending grade

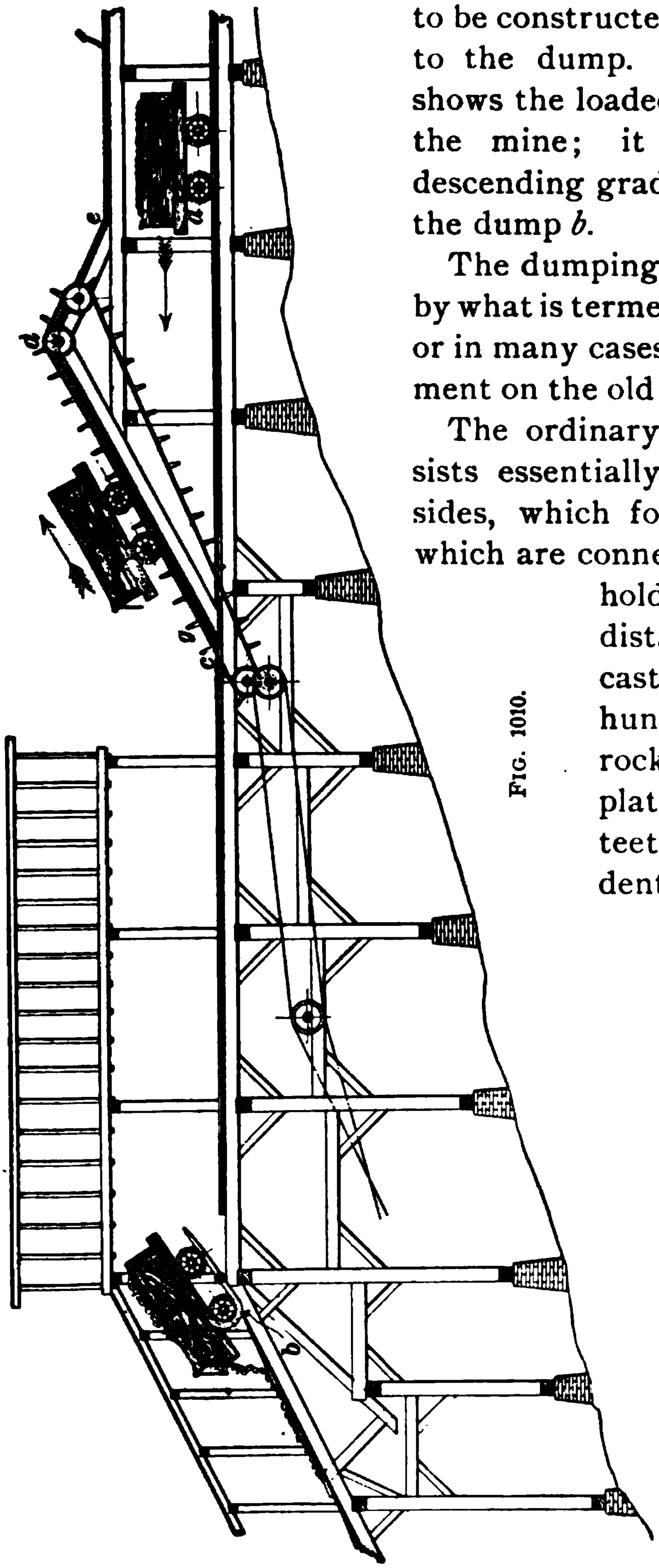


FIG. 1010.

to be constructed from the opening to the dump. In the figure, *a* shows the loaded car coming from the mine; it continues on a descending grade until it reaches the dump *b*.

The dumping is here performed by what is termed a **cradle dump,** or in many cases by some improvement on the old style cradle dump.

The ordinary cradle dump consists essentially of two cast-iron sides, which form the track and which are connected by bars that hold them the proper distance apart. The cast-iron sides are hung on adjustable rockers, resting on plates provided with teeth that fit into indentations on the rocker, to prevent it from slipping. As the car runs upon the dump and strikes the horns *b*, which curve upwards, a man or boy knocks up the door-latch, or fastening. As soon as all the coal has run out of the car, the dump is pulled down by a lever, and the car runs off.

From the dump the empty car runs to the foot of the hoist *c*. This hoist is fitted up with an endless chain. To the chain at intervals are attached projecting pieces, or grips *g*, which take hold of the axle of the car. The power is transmitted to the hoist by means of a wire or hemp rope, connected in some way with the breaker engine.

The car in running over the knuckle *d* is freed from the chain, and in running down the short incline *e* it acquires a certain momentum; this, together with the grade of the empty track *f*, carries the empty car back to the mouth of the opening.

Where the opening is a slope, some such safety arrangement as shown in Fig. 996 is used, so that the cars, in returning from the dump, will not run into the slope.

2813. Figs. 1011 and 1012 show the method of getting the coal to the top of the breaker by means of an inclined plane and a barney.

This method is generally adopted where, by using an inclined plane, a descending grade can be had for the loaded cars from the opening to the foot of the plane, and also a descending grade from the bridge at the foot of the plane for the empty cars to return to the opening. Or, it is frequently used where the coal is brought to the foot of the plane by a small locomotive or by an electric motor from some opening located at a considerable distance from the breaker.

The great success of this method in handling the coal lies in the facility with which the cars are handled at the foot of the plane, and in the rapidity with which they are handled on the dump.

The essential parts of this arrangement are the swinging bridge *A*, the barney *B*, and the arrangement of the tracks at the top and bottom of the plane.

2814. Fig. 1011 shows the arrangement at the bottom of the plane; *C* is a section through *Y Y*, showing the arrangement of the tracks; *D* is a section through *X X*, showing the arrangement of the swinging bridge which is used

From the dump the empty car runs to the foot of the hoist *c*. This hoist is fitted up with an endless chain. To the chain at intervals are attached projecting pieces, or grips *g*, which take hold of the axle of the car. The power is transmitted to the hoist by means of a wire or hemp rope, connected in some way with the breaker engine.

The car in running over the knuckle *d* is freed from the chain, and in running down the short incline *e* it acquires a certain momentum; this, together with the grade of the empty track *f*, carries the empty car back to the mouth of the opening.

Where the opening is a slope, some such safety arrangement as shown in Fig. 996 is used, so that the cars, in returning from the dump, will not run into the slope.

2813. Figs. 1011 and 1012 show the method of getting the coal to the top of the breaker by means of an inclined plane and a barney.

This method is generally adopted where, by using an inclined plane, a descending grade can be had for the loaded cars from the opening to the foot of the plane, and also a descending grade from the bridge at the foot of the plane for the empty cars to return to the opening. Or, it is frequently used where the coal is brought to the foot of the plane by a small locomotive or by an electric motor from some opening located at a considerable distance from the breaker.

The great success of this method in handling the coal lies in the facility with which the cars are handled at the foot of the plane, and in the rapidity with which they are handled on the dump.

The essential parts of this arrangement are the swinging bridge *A*, the barney *B*, and the arrangement of the tracks at the top and bottom of the plane.

2814. Fig. 1011 shows the arrangement at the bottom of the plane; *C* is a section through *Y Y*, showing the arrangement of the tracks; *D* is a section through *X X*, showing the arrangement of the swinging bridge which is used

to take the empty cars off the plane; *E* is a plan of the swinging bridge, showing the arrangement for opening and closing the bridge. This bridge consists of two trusses, having bearings at *c* and *d*, so that by swinging around them the bridge is opened and closed.

The dotted lines in *D* show the position of the bridge when open, ready for the loaded car to ascend the plane; the full lines show the position of the bridge when it is closed, shown also in the plan *E*; ready for the empty car to run over the bridge in descending the plane.

The hoisting rope is attached permanently to the barney *B*. This barney is made in two parts, *e* and *f*, united by the hinge *g*. The part *f* carries four wheels, which run over a narrow-gauge track located within the track that the mine-car runs over, as shown at *s* in the cross-section *C*.

The part *e* of the barney which is known as the **pusher** carries two small wheels *h*, one on each side, known as the pusher wheels. The part *i* is known as the **check horn,** which is used to keep the mine-car in position during the operation of dumping. The barney is made narrower than the car, and passes between the sides of the swinging bridge and between the rails of the mine-car track. The barney track is continuous and unbroken from the barney pit to the dump at the top of the breaker.

The mine-car *j*, as it comes from the mine, is run to the foot of the plane and placed in position, as shown. The barney, from its position in the pit, is back of the loaded car and pushes it up the plane. In going up, the loaded car finds the bridge open; but when the barney reaches the lever *k*, it pushes forward the levers *l* and *m*, and closes the bridge, so that it will be ready for the empty car when it descends.

When the barney descends it finds the bridge closed, and when it reaches the lever *n* its axle pushes forward the levers *o* and *p* by means of the lever *n*, and thus opens the bridge for the loaded car to ascend the plane. The dotted levers marked *n n*, *p p*, and *o o* are the same levers, but shown in different positions. The two pusher wheels *h* of

the barney, as it descends into the pit, lift the latches *q*, which are hinged at their upper end and fall freely, so that when the engine is reversed and the barney is hoisted these latches catch the wheels *h* and force the pusher *e* open. As the wheels are obliged to follow the track *r*, the pusher comes in contact with the car *j* and forces it forward and then up the plane. It will be observed that the larger wheels of the barney, while in the pit, have a rail *s'* above as well as the rail *s* below them; this is to prevent the wheels from rising, and the barney from getting off the track.

The opening and closing of the swinging bridge are performed, as already stated, by the levers *k* and *n*, in conjunction with the levers *m* and *p*. There are rods *v* and *w*, leading from the levers *m* and *p* to the lever *x*, which works on a pivot, as shown, supported by the piece of strap iron *y* which is bolted to the stringers of the inclined plane. At the end of the lever *x* are the levers *z* and *z'*, made, as shown, with the different joints. These levers connect the trusses of the swinging bridge with the lever *x*. From this plan it can be readily understood how the operation of opening and closing is performed by operating the rods *v* and *w*.

The weight *a*, in connection with the lever *b*, is used as a counterweight to balance the bridge-shifting mechanism.

2815. Fig. 1012 shows how the dumping is performed by means of the barney at the top of the breaker. The mine-car *j* follows the track *t*, and as soon as the wheels of the car *j* strike the horns *u* the car is stopped, and the barney continuing to move raises the back end of the car until it assumes the position shown in the figure, the body of the car turning around the front axle.

Where the above barney and method of dumping are in use, it is customary to use a drum having a friction-clutch, so that the engine is used only for hoisting the loaded car, the empty car being loaded by means of a brake. Where this is done the engine is always kept running in the one

direction, and by simply applying the clutch the desired result is obtained. At the head of the plane a man is employed, whose duty it is to take the tickets from the car, knock open the latches, and see that the car is entirely rid

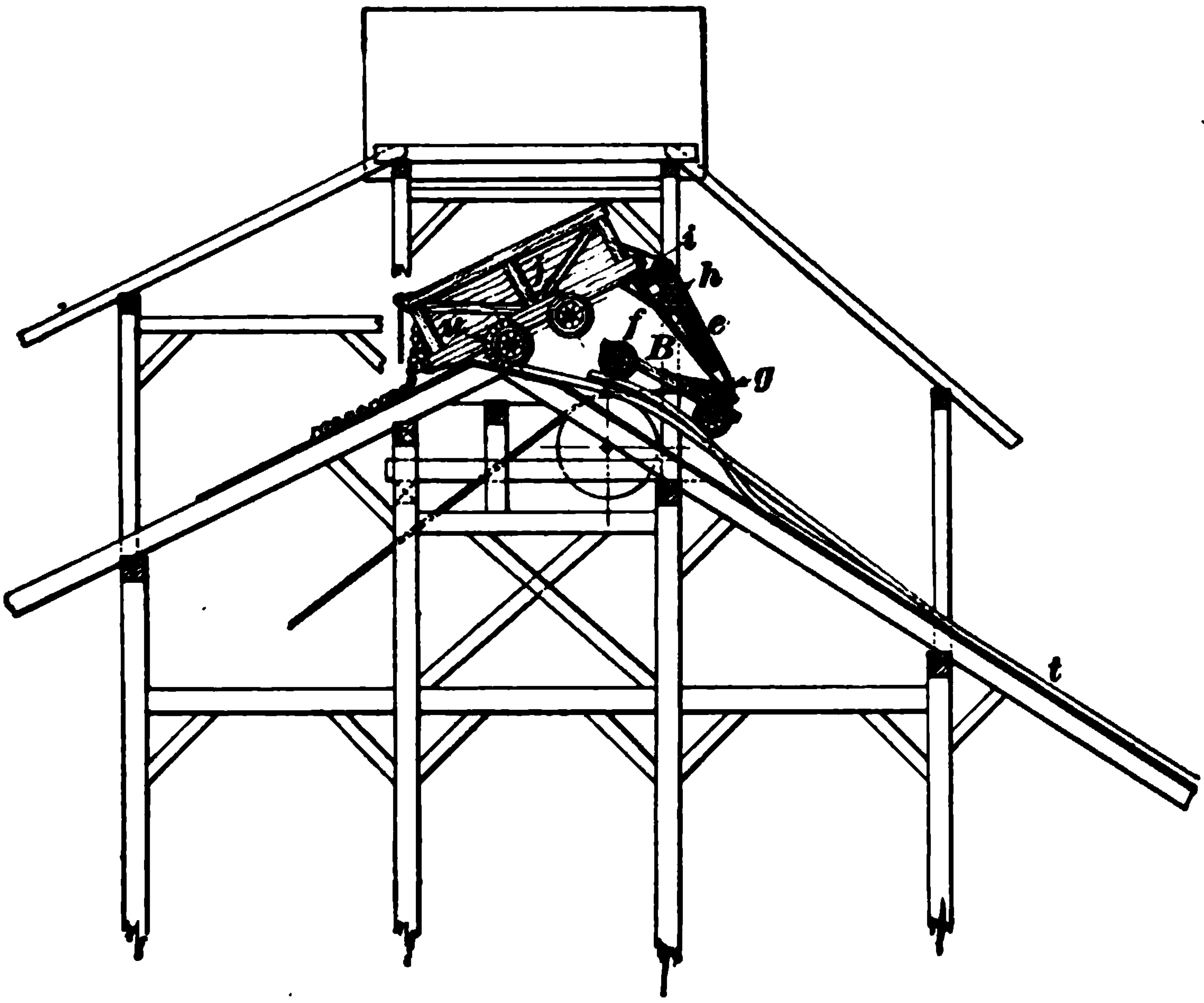


FIG. 1012.

of its contents before being allowed to return to the mine In most cases, however, the door of the car is opened automatically.

2816. Fig. 1013 shows the method of dumping where the breaker is erected in line with the slope or where there is a continuation of the slope through an open trestle. The hoisting-engine in this case is located either in line with the slope at some point back of the breaker or in the lower part of the breaker. In this case the rope runs through the breaker, as shown, the engine being located in the lower part of the breaker. In this method of dumping, the rope is attached to the mine-car by what is termed a **spreader.** The spreader consists of two pieces of wire rope or chain

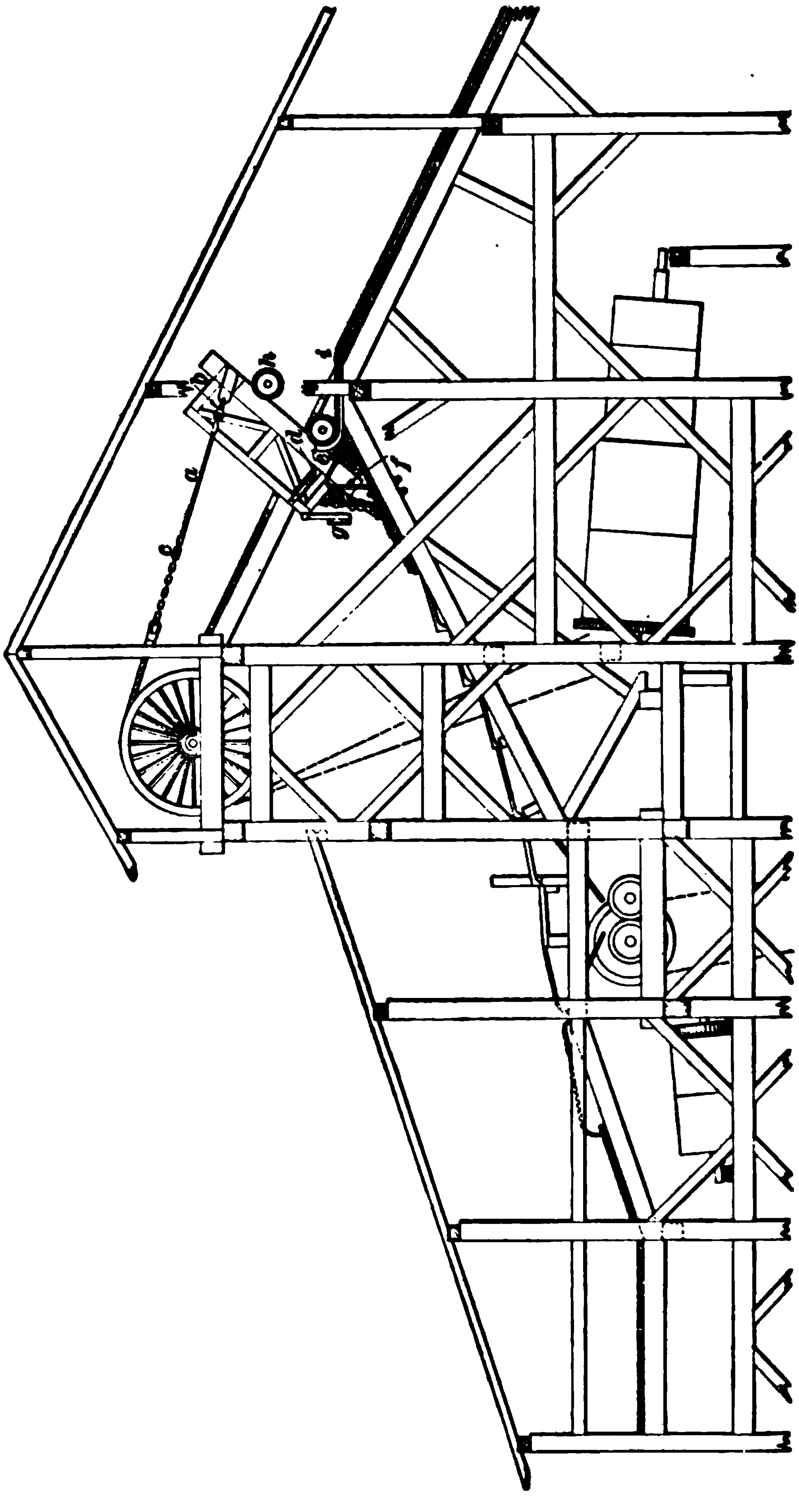


FIG. 1018.

a, with two hooks *b*. At *c* there is a **spreader stick,** which is used to keep the wire ropes spread.

This spreader stick is usually made of a piece of 1½-inch to 1¾-inch gas-pipe. A piece of wood of the dimensions required to resist the strain is entirely too heavy. The cars for this method of dumping are built so that the hooks *b* of the spreader can be attached to the side at the rear end of the car, as shown. At *d* there is a carrying hook to hold up the spreader from the wheels of the car while the car is ascending and descending the plane. The wheels of the car, in ascending the plane, strike the horns *e*; the engineer in charge of the engine continues to hoist, so that the body of the car turns around the front axle into the position shown, a constant pull being maintained upon the rope.

This method of dumping is known at the mine as the **forward dump,** in distinction from the one where the door is placed at the rear in ascending the plane.

The arrangement *f* is used for opening the latches *g* on the car. When the wheels of the car strike the horns *e*, the latches of the car are directly above *f*, and as the car is raised the part *g* presses down on *f*, thus relieving the door.

In designing the above style of dump, the main point is to get the hind wheels *h* of the mine-car to strike the short knuckle *i*, so that when the car is lowered it will move down the plane without the assistance of any one to start it off.

2817. Fig. 1014 shows the method of dumping and getting the coal directly into the breaker from a slope that is on a very heavy pitch. Generally, where the slope is pitching above 60° it is not economical to hoist the coal to the surface in mine-cars, but to accomplish it by the use of gunboats, or, as they are sometimes called, *monitors.*

The following reasons are given for this: Much labor is saved in switching, running the cars on and off the slope, attaching and detaching the rope at the bottom, and the same operations at the top. When hoisting up a slope where the pitch is heavy, the strain on the car is very great; this tends to cause accidents, and makes it necessary to have cars

of better construction than would otherwise be necessary if the pitch were lighter. To prevent the coal from falling out of the cars, coverings are attached to them, so that when they reach the foot of the slope these coverings, or doors, are placed in position. This operation requires time and somewhat delays the steady movement that is always found necessary in order to do good work at the foot of the slope.

In some cases a number of cross rails are put on top of the cars, which render them very difficult to load, and also

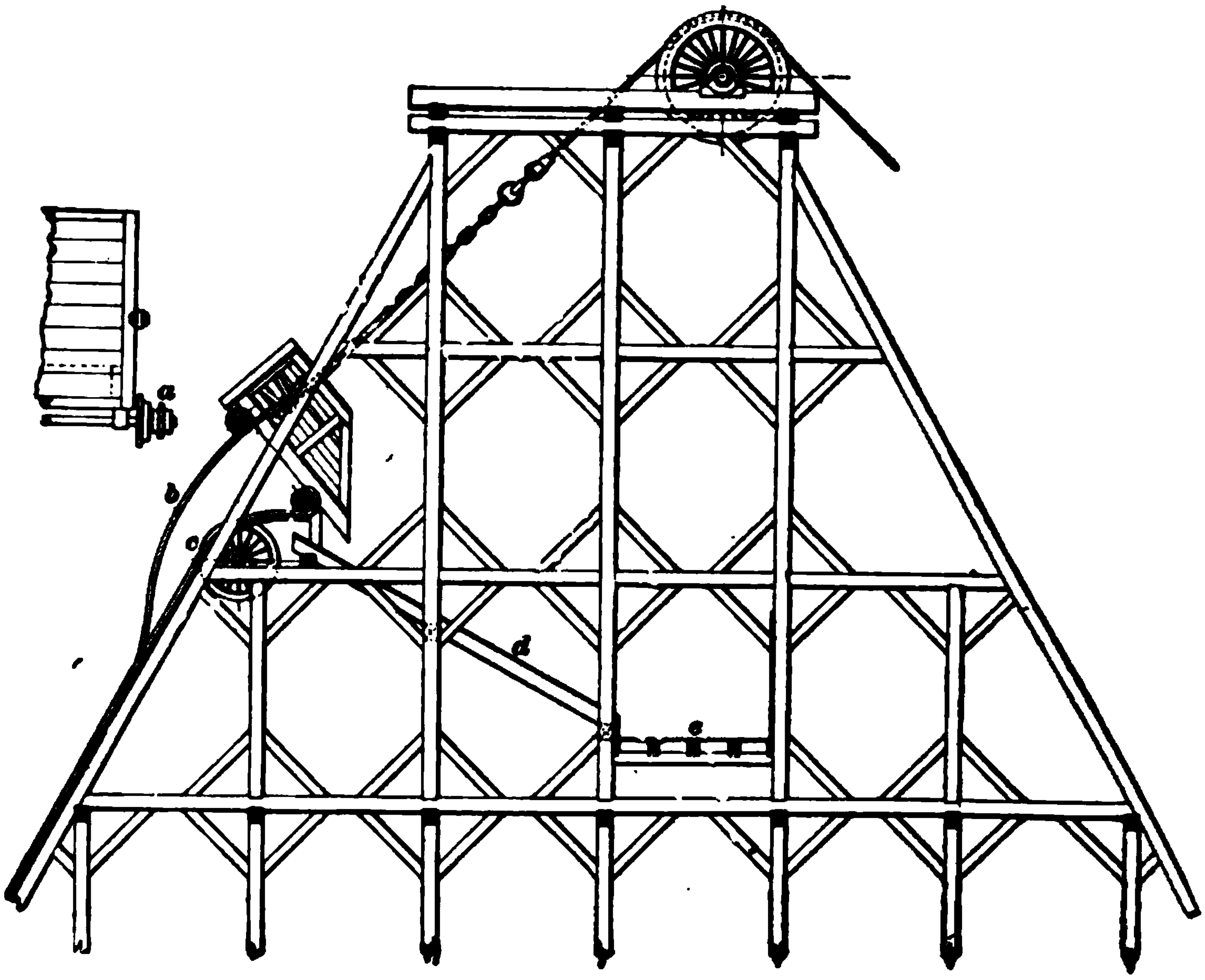


FIG. 1014.

prevent them from being properly filled. For these reasons the gunboat is used on heavy pitches. Where the gunboat is in use the mine-cars circulate only in the underground workings, their contents being dumped into the gunboat at the bottom of the slope or at any other convenient point upon it.

2818. The gunboat is a self-dumping car, very solidly constructed, open at one end. It has a capacity of two or three ordinary mine-cars. The figure shows the side and

the part of an end elevation of a gunboat. The one shown is built of oak timber, lined inside with heavy sheet iron. In many cases the gunboat is built throughout with boiler iron. The wheels are shrunk upon the axles, which are seated in specially designed pedestals.

On the axle at the rear end of the gunboat a wheel *a* is placed which works loosely upon the axle; having a wider gauge, it strikes the auxiliary track *b* just before the front wheels of the gunboat reach the knuckle *c*.

The same object is sometimes accomplished by making the hind wheels of a very much broader tread than the front wheels.

The auxiliary track *b* is a heavy rail bent to the proper curve and securely supported and fastened.

Where gunboats are in use it is frequently the case that quite high towers, as shown in the figure, are necessary, which require considerable bracing to withstand the great strain to which they are subjected by the great weight that is raised at one time. The breaker in this case is erected at the side of the tower, the coal being first dumped into the inclined chute *d*, and from there conveyed into the chute *e*, which is at right angles to the chute *d*.

The bars over which the coal then runs to remove the finer sizes are arranged in a chute parallel to the chute *d*, or very nearly so.

2819. On slopes with very heavy pitches, where the mine-cars are brought to the surface, what is known as the **slope transfer carriage** is used, a side view of which is shown in Fig. 1015. The mine-car *A* is hoisted with a triangularly shaped carriage whose frame *B B B* is built at an angle to suit the slope *C C C*, and on top of which the car *A* rests while being hoisted.

The car is held in place by a drop platform *D*, an arrangement which allows that portion of the platform of the carriage underneath the wheels of the car to sink into the carriage a sufficient depth to keep the car from running off. This drop platform is a separate frame, with the cross-pieces

E, *E* a little longer than the wheel base upon which the car stands. These cross-pieces project at least 4 inches over the sides of the carriage, so that when the catches are put in at the top or bottom of the slope they catch the parts *E*, *E* of the drop platform, and the cage settling back brings the top of the safety blocks *F*, *F*, the cross-pieces *E*, *E*, together with the rails *r* of the carriage, and the drop platform flush

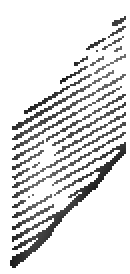


FIG. 1016.

with each other, thus allowing the car to be run on and off. When the rope lifts the carriage off the catches, the drop frame, which is fitted inside the main frame in guides, sinks down the required distance, causing the car to stand in the recess, effectually blocking it, and preventing it from running off the carriage or against the slope timbers during its transmission up and down the slope.

In some cases these carriages run to the top of the breaker, where the car runs direct from the carriage to the dump; or the mine-car is taken off at the mouth of the slope, and the cars reach the top of the breaker by some one of the methods that have been or will be described.

2820. Fig. 1016 shows the arrangement for getting the coal to the top of the breaker by means of a link-belt hoist.

This hoist is in use where the loaded cars from a shaft opening run by gravity to the foot of the inclined plane *a*, where they are fed on either side *b* or *c* of the plane. The empties are returned by a third track *d*, located in the center of the plane, as shown. In order that the empties can be returned to the shaft opening by gravity, they are taken off the plane by means of an overhead trestle *e*.

This hoist is driven by the breaker engine, the gearing being so arranged that the hoist can be stopped and started without interfering with the running of the engine.

The car, in running to the foot of the plane, is **scotched**, which is a term used at the mines to denote a method of hold-

FIG. 1017.

ing the car in the one position. Here it remains until one of the projecting attachments on the chain catches hold of

the mine-car axle and conducts it up the plane. These projections, or grips, are attached to the chain, and located such a distance apart as to regulate the feeding of the cars to the dump in accordance with the time required in dumping.

The figure does not show the arrangement of any safety attachments, but at the mines where a chain hoist is in use a safety device is generally located on the plane at every 10 or 12 feet, to prevent the loaded cars from descending and doing any damage in case of an accident to the chain.

2821. Fig. 1017 shows a very cheap but efficient system of blocks used in connection with chain hoisting. The blocks are made of oak, shod with ¼-inch iron on the side towards the rail. The blocks are fastened to the plane at their lower ends *a*. This fastening serves as a hinge. The block is further supported by the piece *b*, which acts as a brace. The block *c* is kept in position on the rail by the round green oak stick *d*, which is very elastic and acts as a spring. It is fastened to the block by means of a staple *e*, and to the plane by the staple *f*. As the car goes up the plane, the wheels of the car push the block *c* aside; after it has passed, the block resumes its position on the rail, on account of the tension of the spring pole *d*. The cars, after being dumped, are returned over the empty track *g*, which has a descending grade to the mine opening.

MACHINERY USED IN THE PREPARATION OF COAL.

CLASSIFICATION.

2822. The machinery used in the preparation of anthracite coal is divided into the following classes:

Machinery for sizing the coal.
Machinery for breaking the coal.
Machinery for separating the slate from the coal.
Machinery used for conveying the coal into the breaker.

MACHINERY FOR SIZING THE COAL.

2823. This may be divided into two classes: (1) fixed or movable bars and (2) fixed or movable screens.

In the first class the openings through which the coal falls are much longer than they are wide, while in the second class they are nearly square.

In special cases the first class may be used to take out dust or fine coal, otherwise it is seldom employed, except for large coal, or when exact sizing is not important. The reason is that long flat pieces fall out with the cubical pieces of much smaller dimensions, rendering the coal thus sized very irregular in form.

2824. There are three types of the first class in common use:

1. Bars supported at both ends, which are either fixed or adjustable.
2. Finger-bars, supported at one end.
3. Oscillating bars.

2825. 1. The **bars supported at both ends** are either fixed or adjustable, the former being in more general use than any other type.

One great object to be accomplished in screening, whatever the type of bars, is to get the coal to run freely over them, so that the bars will not become clogged up. Bars that are made flat do not accomplish this object as thoroughly as pointed or rounded ones, because on a flat bar the fine fragments have no special tendency to run into the openings between the bars.

Bars with a rounded head are in more general use in the anthracite region than any other form. The one shown at *A*, in Fig. 1018, is a fixed bar supported at both ends. This type is made of cast iron, placed and supported as shown at *B*. The top of the bar is cylindrical and projects beyond the web which supports it, so that any lump which passes through the upper part will fall freely without jamming. This type of bar is usually made about 4 feet long.

The adjustable bar supported at both ends is, as the name

implies, one whose position can be adjusted. The coal to be sized is made to slide longitudinally over it.

The ends of the bars are made V-shaped, to fit into similar grooves on the transverse pieces by which they are supported. The bars can be placed at any required distance from each other, the usual opening being 3½ to 4 inches.

The **finger-bars** shown at *C* are an improvement upon the bars just described and those in ordinary use.

The lower end *f* of the bar is entirely free, and the bars are narrower at *f* than at the upper end *g*, so that should

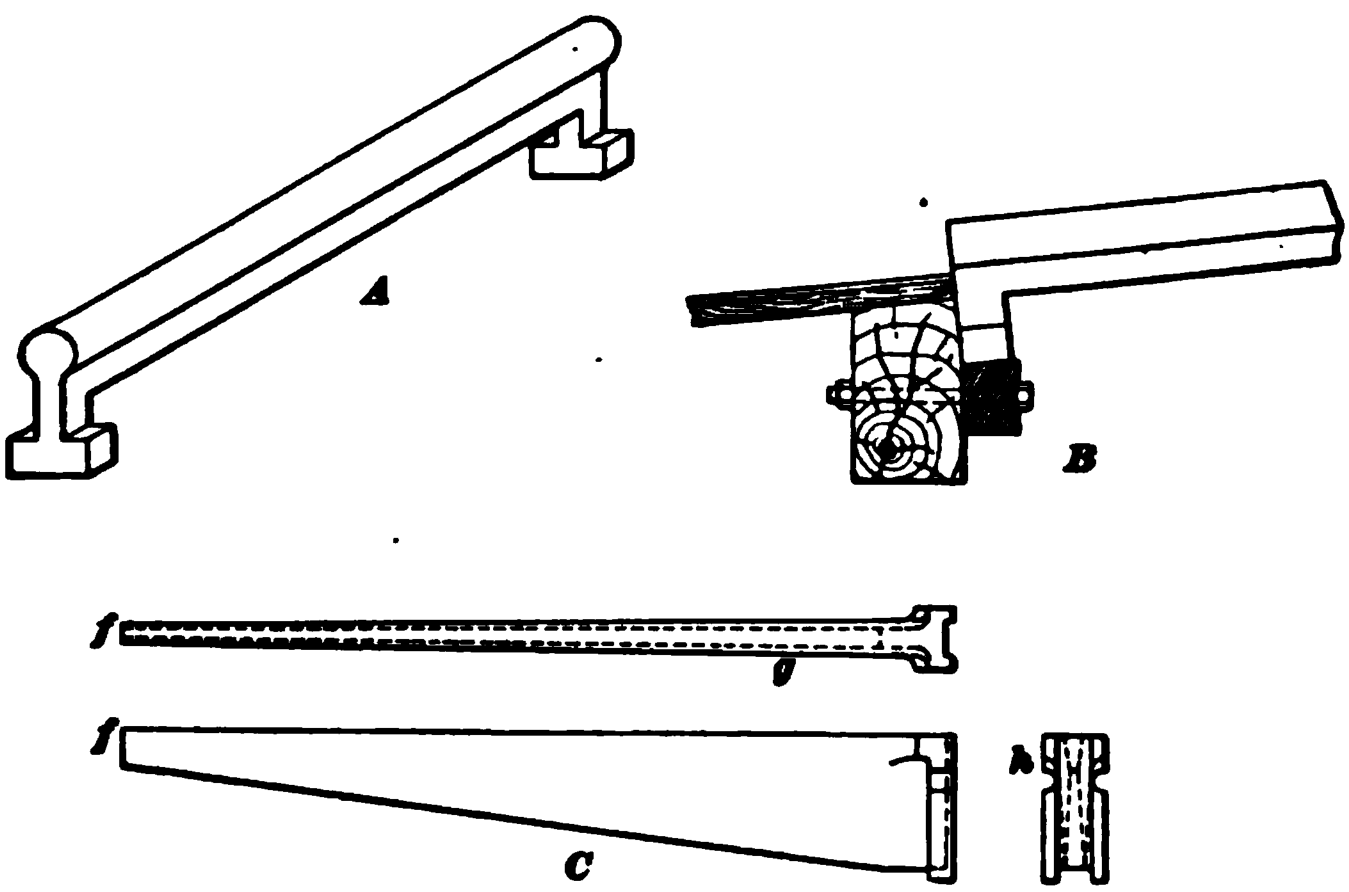


FIG. 1018.

any lump become wedged it is likely to be loosened by the first lump which strikes it.

In the vertical edges of the upper end of the bars are two half holes *h*, by which they are bolted to the beam or bearings.

2826. Movable, or **oscillating, bars** consist of two frames, each carrying a set of narrow bars, placed sufficiently far apart to allow coal of the required size to pass between the bars of each pair. The bars are oscillated back and forth by eccentrics on the main driving-shaft, which are so connected with the bars that the motion of the latter is approximately horizontal. The throw given them is about

3 inches. On the main or driving-shaft there are two eccentrics, placed 180° apart.

The coal fed upon this apparatus at one end will be slowly transported to the other end when the frame is set horizontally, or even if inclined at a slight angle. At the same time the coal is slightly jarred, the dust and dirt shaken off, and all the small pieces are sure to fall through into the hopper below.

Reciprocating screen bars of this description undoubtedly reduce the amount of labor to be performed by the men on the platform, and at the same time reduce the height of the breaker.

2827. Fixed screens consist simply of an inclined plane, formed either of woven-wire screens or punched or cast plates, with round, square, oblong, or other shaped holes. The coal in this case is allowed to slowly slide or roll down this plane by gravity.

The larger pieces pass over and the smaller fall through it. By placing several screens with openings of decreasing size underneath one another, or a series with openings of increasing size in the same chute following one another, any desired number of sizes can be made. While the coal is being handled or moved in the breaker, a certain proportion of small pieces always breaks off, so that when the coal is loaded from the pockets into the cars it is necessary to take out this finer coal. This is done by allowing the coal to pass over a set of bars just before entering the railroad-car. When the fixed screen is used for this purpose it is known as the **lip screen.**

2828. Movable screens are among the most important parts of a breaker. They are of two types. In the first type the screening surface forms a cylinder and revolves about its axis; in the other type the screening surface is approximately horizontal.

2829. Before discussing the construction of screens and methods employed, it is first necessary to know the

sizes of coal that are to be prepared in the breaker. The sizes generally prepared are as follows, arranged in order, commencing with the largest size first: **Lump, steamboat, broken, egg, stove, chestnut, pea, buckwheat,** and **rice.** Lump coal is prepared on the platform, and all the other sizes are prepared by sizing in different screens, known as the steamboat, broken, egg, stove, chestnut, pea, buckwheat, and rice screens.

2830. A breaker generally has a screen known as the slate-picker screen, and there are others known as the *main* and *counter*, or *mud*, screens.

Screens are made **single** and **double jacketed,** and derive their names from the size of the coal which they prepare in the **first jacket,** or in other work. Screens are always designated by the size of coal that comes out of the end; for example, the steamboat-coal in a steamboat screen comes out of the end, while all smaller sizes fall through the meshes and pass on to some other screen.

The term **main screen** designates a screen preparing several sizes; for example, the main screen usually prepares egg, stove, and chestnut, although it would not be wrong to term the above an egg-coal screen.

The **mud screen** is the name given to the one that takes the coal coming from the main bars, or, as they are sometimes called, the **platform bars.** This screen is also sometimes called the **counter screen.** The term *counter screen* always refers to a screen that is located above the main screen or other sizing screen, through which the coal is run and partly sized before entering another screen for the final sizing. This is done in order to get the slate out of the larger sizes of coal by hand picking.

2831. In many cases breakers have what are known as **counter, mud, chestnut, pea, buckwheat,** and **rice screens.** These are located below the mud screens and prepare the coal that comes from the mud screens directly for the pockets for shipment, or for use at the mines to generate steam. This coal is not mixed with the coal that

comes from the main crushers and that coming from the prepared-coal rolls.

Slate-picker screens are cylindrical screens jacketed with cast-iron segments, provided with narrow slits through

which the flat pieces of slate fall, but through which the coal (not being flat) can not pass.

2832. Fig. 1019 shows a main screen used in the anthracite region, which is usually from 16 feet to 30 feet in length. The diameter of the first jacket *a* is usually from 5 to 6 feet, while the diameter of the double-jacketed part *b* is usually from 6½ to 8 feet. These screens are set on an inclination of ¾ inch to 1 inch to the foot, so that the coal will travel slowly from one end to the other. They are run at from 8 to 10 revolutions per minute.

These screens are constructed of a number of cast or wrought iron spiders set at intervals of from 3 to 5 feet apart, on wooden or wrought-iron shafts, although of late years Phœnix columns are being used for the shafts.

The spiders are made up of a cast-iron hub *c* having four wrought-iron arms *d* bolted to it. The arms are either riveted or bolted to the ring or circle *e*, and the hub is keyed to the shaft *f*.

The jackets *a* and *b* may consist of either a series of wire, cast-iron, or wrought-iron, or steel punched gratings of the proper mesh, called **segments.** In the figure wire segments are shown. The meshes of a segment always refer to the size of the openings that permit the coal to drop through. The distance between any two spiders is referred to as the 1st set or row of segments, 2d set or row of segments, etc.; hence, by increasing the number of spiders the number of sets or rows of segments is increased. In speaking of a screen, reference is always made to the back end and front end; the back end refers to the end where the coal is fed to the screen, and the front end is the end where the coal that does not drop through the meshes is delivered. But in referring to the sets or rows of segments, it is customary to speak of the 1st, 2d, etc., set or row of segments, commencing to number from the back end towards the front end.

The segments are fastened to the rings *e* by six bolts, three at each end. As shown in the figure, the segments

are so arranged that the joint at any two, as *g*, will come in the center of a segment in another set or row as *h*. In this manner the screen as a whole is firmly bound together.

The jacket *b* is spoken of as the double or outside jacket. The meshes of the segments of this jacket are always smaller than those of the segments of the first jacket, or those directly under it.

The second or last row of segments *b* on the outside jacket is made quite different from any other of the segments shown.

A small opening from 6 inches to 9 inches is made in this segment, so that the coal that does not pass through the meshes of the segments composing the outside jacket can drop out before coming to the end of the set of segments *b*. This is done so as not to interfere with the coal that drops through the meshes in the set *a*.

The outside jacket is supported by pieces of gas-pipe *i*, which envelop the arms of the spider projecting through the circle of the first jacket. The circle for the first jacket over which the double jacket is located is supported by collars provided on the spider arms. That part of the arms which projects beyond the gas-pipe and outside circle is either riveted or threaded and fitted with a nut.

2833. Two methods of driving screens are in common use:

1. By bevel-gears on the screen shaft, usually at the front end of the screen.

2. By spur-gears on the periphery of the screen, at its back end, as shown in Fig. 1019. This large gear is always cast in one piece, and also contains the circles for the inside and outside jackets.

The pinion *j* is usually placed under the screen, and its shaft is always horizontal. This, of course, gives a greater bearing on one side of the teeth than on the other, and in time one side becomes greatly worn, while the other side is comparatively unworn. The pinion is then changed end for end.

The screen is supported at the back end by a hanger. The one shown is made up of three parts. The part *k* is bolted to one of the main stringers in the breaker. The part *l* contains a small saucer-shaped depression which supports a correspondingly shaped projection of the bearing *m*. The one fits into the other, thus allowing a free and easy movement for the screen shaft. The bearing contains two of these saucer-shaped projections, so that as it wears away it can be inverted. In many cases the parts *k* and *l* are cast in one piece, with an opening in the back, so that the bearing can be inserted.

The segments at the back end of the screen, or the end into which the coal is fed, have the smallest mesh, and those at the front end have the largest mesh. Wire segments make the finest separation of coal; but where water is used the wires soon slip, which increases the size of some meshes and decreases the size of others. Cast-iron segments are generally used for the steamboat and broken coal screens, while punched wrought-iron segments are used for the rice-coal screens, although in many instances the latter are used for all types of screens.

2834. The coal that enters the screen shown in Fig. 1019 contains every size below that of broken coal, which has first been separated by a broken-coal screen. The first two sets of segments remove the dirt, rice, buckwheat, pea, and chestnut. These sizes enter the double-jacketed part. All sizes below chestnut drop out of this jacketed part, while the chestnut comes out at the end and passes to a slate-picker screen, or to a screen known as the chestnut-coal screen, which is generally double-jacketed and carries a row of slate-picker segments. This screen is usually 4 feet in diameter, with an inclination of $\frac{3}{4}$ of an inch to the foot, and runs at the rate of 15 revolutions per minute. The double-jacketed part is used to take out the pea coal that may have remained in the chestnut. The slate-picker segments in this screen take out the flat slate. The chestnut coal coming out of the front end of the screen passes on either to a jig or to the coal pocket.

The coal that dropped through the meshes of this double jacket of the main screen passes on to a pea-coal screen. This is also a double-jacketed screen. The first jacket is usually 4 feet in diameter and the jacketed portion 4 feet 6 inches. This screen has usually a $\frac{3}{4}''$ to $\frac{7}{8}''$ pitch, and runs at the rate of 15 to 20 revolutions per minute. The pea coal comes out of the end of the first jacket, and what drops through the meshes is buckwheat, rice, and culm; the buckwheat comes out of the end of the double jacket, while the rice and culm pass through the meshes and are carried on to the rice-coal screen. The rice-coal screen is a single-jacketed screen, usually about 12 feet long, 4 feet in diameter, and has an inclination of $\frac{1}{2}$ inch to the foot; it runs at the rate of 13 revolutions per minute. The rice comes out of the end, while the culm passes through the meshes and out into the culm pocket. The chestnut, pea, and rice screens are usually run by bevel-gears on the screen shaft at the front end of the screen.

The coal that comes through the single-jacketed portion *a* of this main screen is known as stove coal, while that coming out of the end is egg coal. The stove coal is either carried to a set of jigs or run over inclined chutes, known as the **picking chutes,** the egg coal going through a similar operation.

2835. The meshes for the different segments used in the sizing of coal are given in the following table:

Culm passes through	$\frac{3}{16}$-inch mesh (round).	
Rice passes over	$\frac{3}{16}$-inch mesh and through	$\frac{3}{8}$-in. sq.
Buckwheat passes over	$\frac{3}{8}$-inch mesh and through	$\frac{1}{2}$-in. sq.
Pea passes over	$\frac{1}{2}$-inch mesh and through	$\frac{3}{4}$-in. sq.
Chestnut passes over	$\frac{3}{4}$-inch mesh and through	$1\frac{3}{8}$-in. sq.
Stove passes over	$1\frac{3}{8}$-inch mesh and through	2-in. sq.
Egg passes over	2-inch mesh and through	$2\frac{3}{4}$-in. sq.
Broken passes over	$2\frac{3}{4}$-inch mesh and through	$3\frac{1}{2}$-in. sq.
Steamboat passes over	$3\frac{1}{2}$-inch mesh and out end of screen.	

Lump passes over platform bars that are set from $3\frac{1}{2}$ to 5 inches apart.

2836. In Fig. 1020 *A* and *B* show a cross-section and side view of the part of a main screen where a wooden shaft *a* is used.

From this figure it is seen that the parts *c* and *d* of the spider are made of wrought iron and in one piece. The screen circle *b* is made in one piece and joined by bolting, as shown at *h*. The screen circle *b* is secured to the spider arms *d* either by riveting, as shown in the figure, or by bolting.

The spiders are secured to the screen shaft *a* by a series of wedges driven between the shaft *a* and the web part *c* of the

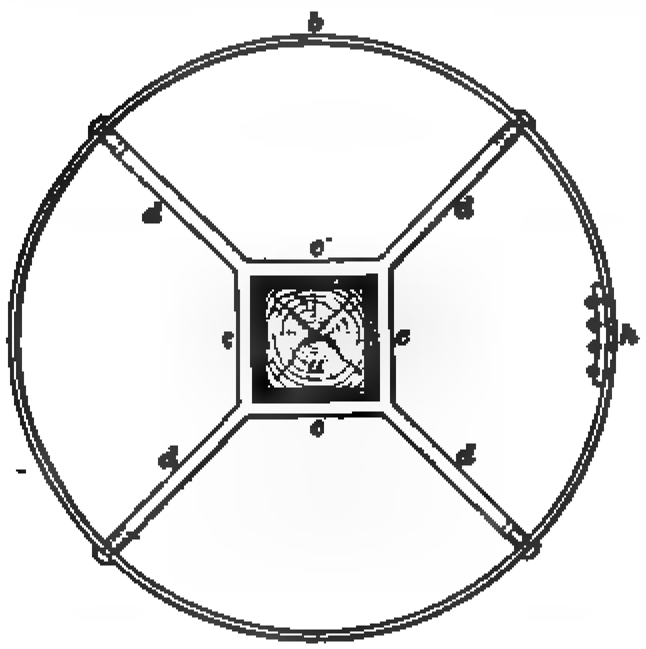


FIG. 1020.

spider. These wedges are single tapered wedges, made of oak or yellow pine, and are driven, as shown in the figure, from *e* and *f*, so that one overlaps the other.

To prevent the screen shaft *a* from wearing out by the continual striking of the coal, it is covered either with light sheet iron *g*, or ¾-inch oak boards. The shaft *a* is generally either white or yellow pine timber, 14 inches × 14 inches in cross-section. The bearings at the front and back end of the screen are specially designed cast-iron pieces, called the **screen's gudgeon.** These gudgeons are fastened to the wooden shaft by a number of large bolts passing through it.

2837. Fig. 1021 shows a **Phoenix column shaft,** which is fast taking the place of the wooden shaft for main

screens, and also the wrought and cast iron shafts for the smaller screens. As shown in the figure, the end bushings are of cast iron, and are riveted in place to the Phœnix column, making a very substantial piece of work. These shafts are very light for their strength, and serve their purpose

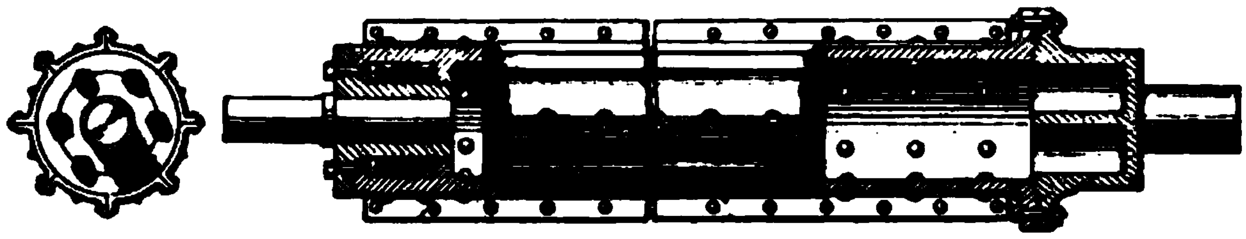


FIG. 1021.

admirably. The spiders used on these shafts are a cast-iron hub with wrought-iron arms, fastened to the shaft by set screws.

2838. The **steamboat screens** are usually from 4 feet to 6 feet in diameter, and never more than 6 feet to 12 feet in length. They have an inclination of $\frac{7}{8}$ inch to the foot, and are run from 10 to 13 revolutions per minute. The screen shafts are wrought iron and solid throughout. The spiders in this screen are cast iron, in one piece, and are keyed to the shaft. The segments are cast iron. This screen takes the coal as it comes from the main rolls or crushers. Both methods in use for driving screens are used to drive this screen, although the spur-gearing on the periphery of the screen and small pinion is used almost exclusively. In many cases the screens are double-jacketed, the outside jacket permitting the sizes below broken to pass through, the broken coming out of the end.

2839. The **broken-coal screen** is a screen that is very rarely double-jacketed. It is from 4 feet to 6 feet in diameter, usually from 9 feet to 12 feet in length, with an inclination of $\frac{7}{8}$ inch to the foot, and is run 10 to 13 revolutions per minute. The segments used on this screen are either cast iron, wire, or punched wrought iron. It is usually driven by the large periphery spur-gear and pinion, in preference to bevel-gears. It takes the coal as it comes from the prepared rolls, often called the monkey rolls.

2840. For cleaning the very smallest sizes of coal, as pea, buckwheat, and rice, a **pentagonal screen** is often used; that is, the screens, instead of being circular, are in the form of a pentagon. These screens have about the same dimensions as the circular screens, but run at a very much higher speed, varying from 30 to 35 revolutions per minute. This form is used to give the coal a greater stirring up. They are always single-jacketed, and are used solely to get rid of the dirt.

2841. When the coal is wet and dirty, washing is necessary to remove the dirt that adheres to each fragment. This is accomplished by a series of small streams or jets of water falling upon the coal in the screens from a perforated pipe or trough above the screen. Or, the trough is set level and the water is fed into it from below from a tank which is so located as to give a certain amount of head to the water, causing a uniform overflow throughout the whole length of the trough. This washing is quite necessary to enable the slate pickers to distinguish the slate and bone, and at the same time it makes the coal more salable.

In order that the screens may do their proper sizing, they should not be overcrowded; for when they are crowded with coal a considerable quantity of one size becomes mixed with that of another. To overcome this, some breakers have automatic screen feeders.

2842. The **capacity of screens** depends upon the amount of screening surface, the inclination, and the speed at which they are run. Increasing the pitch has the same effect on the screening as shortening the screen, but it increases instead of diminishes its capacity. Hence, the screening capacity in large breakers is increased by increasing the pitch of the screens, although to secure thorough screening very long screens are necessary, while at breakers handling a small output the screening may be equally well done by short screens with a very small inclination.

To give some idea of the capacity of screens, the following are the results of tests made in the anthracite region. The

first was that of a main screen (dry) with ¾-inch pitch, 18 feet long. The first jacket was 5 feet in diameter. It also contained a jacket for chestnut, 6½ feet in diameter and 7½ feet in length. The screen was run at a speed of 10 revolutions per minute. The meshes for the outside jacket were ¾ inch. The first jacket had 8 feet of 1⅜-inch meshes, while the remainder was made of 2-inch meshes. The amount of egg coal that came out the end was six tons per hour; the amount of stove coal, ten tons per hour, and of chestnut, nine tons per hour.

The next test was that of a pea-coal screen 4 feet in diameter, 12 feet long, ¾-inch pitch, and running at the rate of 14 revolutions per minute. The screen was a wet one and supplied with an abundance of water. The meshes in the first jacket were ½ inch square. The buckwheat passed over a ⅜-inch mesh and the rice over a ⅛-inch mesh, punched plates being used throughout. Eight tons of pea coal per hour came out the end of the first jacket, and the amount passed through the first jacket was five tons of buckwheat and ten tons of rice coal. The above was calculated at 2,240 pounds to the ton. On account of the water, it was found very difficult to get the amount of culm.

Three tests were made of the above screen, and gave 6, 8, and 10 tons per hour, respectively, of pea coal. It was found that 8 tons per hour was about all the screen could size, for when tested for 10 tons it was quite impossible to size the buckwheat.

2843. In another type of movable screens the screening surface is approximately horizontal, and the motion and action is similar to that of an ordinary hand sieve. Such a screen is shown in Fig. 1022, and is known as the **shaking screen,** or **shaker.** This type of screen is used for sizing small coals; the one shown in the figure being made for sizing pea, buckwheat, and rice.

This screen is moved backwards and forwards by means of the eccentrics, as shown. This motion, combined with the inclination of the screen, causes the coal which is fed on

the highest part of the screen to travel gradually across it. Pea, which is the coarsest coal, comes off first, at *A*; buckwheat second, at *B*; the rice coal coming out over the

Fig. 1022.

end *C*, while the culm passes through the meshes in the bottom.

2844. The shaking, pentagonal, and circular screens work very well for the smaller sizes of coal, where they are comparatively dry, but where they are wet it is very difficult to get out the dirt and make a complete separation. To overcome this a machine has been invented which is known as the **Righter coal washer and separator,** and which is shown in Fig. 1023. Its main feature is a screen box *a*, provided with perforated plates upon both top and bottom, and it has also a slate pocket *b* near its lower end. The screen box is submerged in water and is inclined towards the coal elevator *c*. This screen box is given an up-and-down motion and at the same time a backward-and-forward motion by means of the different eccentrics used, which are driven by the bevel-gearing, as shown.

The coal is fed into the screen box at its upper end *d*, and the motion of the screen box, aided by its downward inclination, causes the coal and slate to work forward; the slate being heavier is deposited in the slate pocket, while the

coal is carried and delivered to the elevator *c*, where it is discharged into a chute, which leads either to a pocket or to a sizing screen. The slide in the bottom of the slate pocket *b* is opened from time to time, as shown, by means of the rods and levers. The bottom of the tank is inclined, which

Fig. 1023.

allows the slate and mud to pass very readily to the elevator *g* which removes it.

2845. Another method that is used for cleaning the smaller coals before sizing is shown in Fig. 1024, which consists of a water-tight tank *a*, filled with water to the line marked *b*. The tank is closed at the top by the perforated plate *c*, having holes to suit the smallest size of coal to be separated. The plate *c* extends up the inclined chute *d*, so as to permit the drainings to get back into the tank *a*. An endless conveyor *e* is used to drag the mingled coal and dirt over the plate *c* and up the inclined chute *d*. The culm containing the coal to be washed is supplied by the chute *f*.

This culm falls into the water, is moved along and thoroughly stirred up and tumbled over and over upon the perforated plate *c* by the conveyor. In this operation the fine dirt sinks through the perforations and falls into the bottom of the tank, while the coal is thoroughly washed and moved

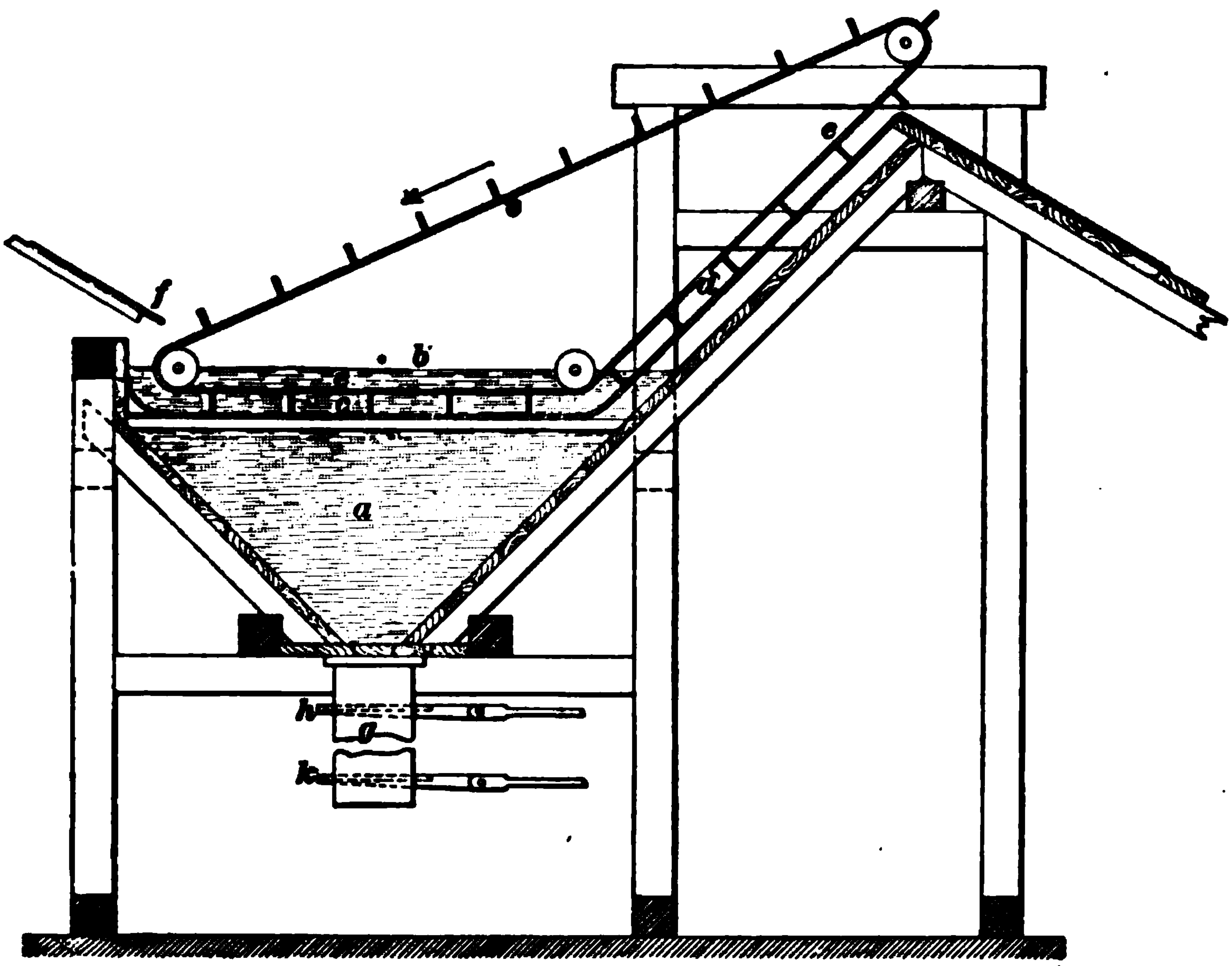


FIG. 1024.

along the bottom and up the inclined chute *d*, to be delivered to screens or other receptacles, as may be desired. The dirt which accumulates in the bottom of the tank is discharged by means of the slush box *g*, which has two wedge-shaped slides *h* and *k*. Upon the upper surface of the slide is a piece of oak wood. The slides move in a casting which has on each side a taper groove, whose wedging action furnishes a water-tight joint.

The discharging is performed by the first opening *h*, *k* remaining closed; the fine mud that was above *h* is then lowered to *k*; *h* is then closed and *k* opened, which discharges the waste without letting the water out of the tank.

2846. Gyrating Screens.—Another type of movable screen in use at some of the collieries in the anthracite region is known as the **gyrating screen.** This screen receives a gyrating motion like the motion a molder gives to his sieve when screening his sand.

The great advantage claimed for this type of screen is that the whole surface of the screen is constantly in action, while in the revolving screen of say 5 feet in diameter only about 8 inches of the 16 feet circumference is at any time in action, unless the screen is overcrowded; the revolving of the screen acts like an elevator, and tends to throw the coal back into the screen.

2847. Fig. 1025 shows a single gyrating screen, which is made of two parts: the upper or screen box and the lower or box bed-plate. The screen box is made up of shelves, varying in number from 2 to 6, depending upon the size of material to be screened. The box is 4 feet wide and 6 feet long, inside measurements, giving 24 square feet of screen surface per shelf. The boxes are made from 1 foot to 2 feet deep. The smaller the size of coal the closer to each other the shelves can be put.

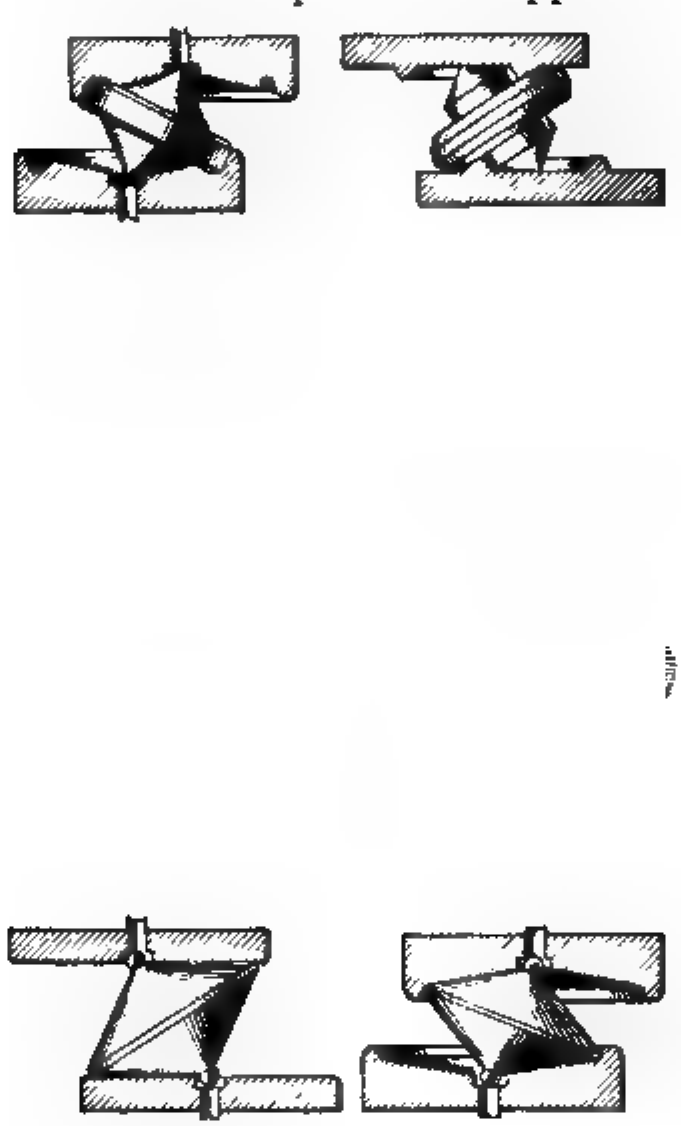


FIG. 1025.

On the shaft shown in the bed-plate a pulley is placed to drive the screen. The large wheel shown between the screen box and the bed-plate is counterweighted, to balance the centrifugal force of

the screen box. Above this wheel is a crank to drive the screen box. As shown in the figure, the screen box rests on four double cones, which are supported by the box bed-plate. The cones roll freely in a prescribed path, and the guiding of the cone is done by ball-and-socket joints at the two points of the cone, as shown in the two lower views, or by one of the two arrangements shown in the two upper figures.

These screens run from 140 to 145 gyrations per minute. The coal is fed at the top of the screen box; the smaller sizes drop through the openings, allowing the larger coals to come off first.

There is no clogging of the holes, as would at first seem likely. The circular form of the holes and the tendency of the pieces of coal to move in a small circle cause the holes to clear themselves without difficulty.

MACHINERY FOR BREAKING THE COAL.

2848. For breaking up the coal two methods are used. The one already referred to, and known at the mines as *chipping*, is where the lumps are large and the pieces of slate attached to them are of such a character as to render it economical. The larger lumps are broken by hand, the men using picks made for that purpose; but by far the larger portion of the breaking is done by **rolls.**

2849. The rolls used in breaking coal are of three kinds: (1) those with pointed teeth; (2) those with the continuous teeth, known as the **corrugated rolls,** and (3) those known as the **saucer rolls,** where the teeth are in the shape of knife-blades.

The coal is generally broken by two or three sets of rolls, as follows:

1. The main rolls or crushers, often called the lump-coal rolls.

2. The prepared-coal rolls, often called the monkey or pony rolls.

3. The bony-coal rolls, used to break up the bony coal.

Some collieries, however, have rolls for breaking lump into

steamer, another for breaking steamer into broken, another for breaking broken into egg, another for breaking egg into stove, a fifth for breaking stove into chestnut, and a sixth for breaking chestnut into pea coal.

2850. In the above operation all sizes are made below the size which is being broken, yet the most economical method is to break any size as nearly as possible into the size immediately below it; in other words, it is more economical to break lump as far as possible into steamer, then break steamer as far as possible into broken, then broken into egg, and so on, each time taking out all the coal below the size to be broken before passing that size through the rolls.

2851. The **main rolls,** or **crushers,** are generally located underneath the main platform. All the coal that does not fall through the main bars is either made into lump coal or passed through the main rolls. Here it is crushed and passes out into the steamboat screen, where the steamboat-coal is taken out.

Fig. 1026 shows the plan *a* and side view *b* of a set of main rolls, or crushers, with pointed teeth. These rolls are plain

Fig. 1026.

cylinders, made either of cast iron or steel casting, with hubs bored and key-seated to receive the shafts.

Formerly the rolls consisted, as they do at many collieries yet, of four cast-iron spiders, two on each shaft. These

spiders are keyed to the shaft and covered with from three to four cast-iron segments, which are bolted to the spiders, each segment having four bolts, two at each end; the teeth are usually the pyramidal form, cast in with the segment. The great trouble with this kind of a roll is that when one or more teeth break off from a segment, it is necessary to throw the whole segment away, it having become useless. However, these segments are an improvement on the old style of rolls that were cast solid.

The latest improvement in rolls is where the teeth are made of steel and inserted in either a cast-iron or steel cylinder, as shown in Fig. 1026. The cylinders are turned in a lathe and the holes are then drilled and reamed to receive the steel teeth. These teeth are made so that they can be removed from the cylinder and sharpened, or new ones inserted.

2852. The steel roller teeth *A* and *B*, Fig. 1027, show the latest types used for anthracite-coal rolls.

The type *A* is known as the **pyramidal tooth.** Teeth of this type are made in different sizes and numbered. When the pyramidal part of the tooth extends beyond the cylinder 2¾ inches, the curved part is 2⅝ inches long, and 1¼ inches wide at the bottom. The part *a* of this tooth projects beyond the cylinder, or shell, of the rolls and aids in the extraction of the tooth after it has become worn.

The type *B* is known as the **hawk bill tooth.** Instead of having the front edge of the tooth bent near the point, it is straight. This type of tooth is four-sided, having the back

FIG. 1027.

of the tooth curved, so that the points of the teeth strike the lumps and draw them into the rolls, splitting them at the same time. These teeth are also made in different sizes and numbered.

Two flat places *a* are provided on this tooth to take hold of when it becomes necessary to extract it.

2853. The bore of the main breaker rolls is generally 8 inches, and the shafts are of wrought iron. The diameter of the fly-wheel and belt pulley and length of driving-shaft are made to meet the requirements of the driving power, as the speed at the points of the roller teeth should be about 1,000 feet per minute. The average sizes of these rolls are from 2 feet 9 inches to 3 feet 6 inches in diameter, and from 3 to 4 feet long. There are five pedestals, only four, however, being shown in Fig. 1026, and they are generally lined with babbitt metal.

The pedestals rest on cast beds, which are provided with adjusting keys, so that the rolls can be set farther apart or closer together, to increase or decrease the quantity of the larger sizes of coal. The teeth of the spur-gears for driving the rolls are usually 3½-inch pitch and 8-inch face. The holding-down bolts are 1½ inches in diameter, and of a length to suit the timbers for which they are used. The rolls, as shown, are enclosed by a cast-iron casing called a **hopper,** so arranged that it can be very readily taken apart.

The larger sizes of coal drop through the meshes of the steamboat screens. The over-supply of steamboat-coal for market and the coal that comes out of the end of the mud screens is run into a pair of rolls known as the prepared-coal rolls, or monkey rolls.

2854. Prepared-coal rolls differ only in dimensions from the main rolls. The usual bore of the prepared-coal rolls is 5¾ inches.

The diameter of the fly-wheel and belt pulley and the length of driving-shaft are made to meet the requirements of the driving power, as the speed at the points of the roller teeth should be about 1,000 ft. per minute.

The teeth of the spur-gearing for driving these rolls are usually $3\frac{1}{2}$-inch pitch and 7-inch face.

These rolls are generally 19 inches in diameter and 29 inches long.

2855. Bony-Coal Rolls.—These rolls are similar in construction to the main and prepared-coal rolls, although of a much smaller size than either, and are used to break up the bony coal that comes from the pickings of broken, egg, and stove coal. They are run at a much higher speed than the main and monkey rolls, the speed at the points of the roller teeth varying from 1,300 to 1,800 feet per minute.

2856. Corrugated Rolls.—Fig. 1028 shows the plan and side view of a set of adjustable corrugated lump-coal and steamboat rolls.

These rolls differ from those already described in the form of their teeth *a*, which are continuous from one end to the other. There are no points. The ends of the teeth are slightly rounded, and the parts doing the work are cast in chills, so as to give greater endurance. The teeth are cast in one piece with the body *b* and hubs *c* of the roll. The hubs are bored and key-seated to receive the shaft *d*. There are different sizes of teeth, according to the size of the coal that is to be made, and also according to whether the rolls are fixed or adjustable.

2857. By **fixed rolls** are meant those that are arranged to break the sizes successively. In such a case it is not necessary to change the distance between the centers of the shafts of the rolls after the proper distance for most economical breaking has once been determined.

The rolls shown in Fig. 1028 have an arrangement for changing the distance between them while they are running. When this change is great, it is necessary to use gear-wheels of greater or smaller diameter.

From the side view, it will be noticed the pedestals *e* are made so as to remain stationary. The frames *f* carrying the bearings for the right roll are planed and rest on cast beds *g*

which contain planed grooves, as shown in the section *A B*, for the correspondingly planed surfaces of *f* to work in. The frames *f* also carry the square nuts *h* on the short bevel-wheel shafts *i*. The bevel-gearing on these shafts gears

FIG. 1028.

with that of *j*, so that by inserting a bar in the hole *k* the right-hand roll can be moved back and forth.

2858. These figures also show a safety device by which breakage of either the teeth, rolls, or gearing is prevented when the rolls draw in material too hard to crush.

The part supporting the bearings rests against an elliptical shell of cast iron *l* inserted in the rectangular opening, as shown in the figure. This elliptical shell is made thick enough to withstand a thrust of several tons, but if subjected to a greater pressure it breaks and allows the rolls to slide farther apart, thus relieving them of all strain. This cast-iron shell is quickly replaced by removing the cap *m*, covering the box in which it is placed. This safety device and form of gearing, by which the distance between the two rolls can be varied at any time, is also used on rolls with pointed teeth.

The **lump-coal rolls** are usually made 26 inches in diameter from tip to tip of teeth, 3 feet long, with 14 teeth. The gearing for these is the same as that used for the pointed-tooth roll.

2859. Corrugated Rolls for Broken Coal.—The diameter of the shaft used is 4½ inches, and of the rolls 16 inches; the length of the rolls is 3 feet, and the number of teeth 20. These rolls are usually run about 205 revolutions per minute. The result sought by the corrugated rolls is to break the lump into two pieces of nearly the same size, and for this reason this type of roll is used to break the one size as nearly as possible into the next size below.

2860. Taper rolls are a modification of the corrugated rolls just described, and are used when a small quantity of a number of different sizes is to be broken up at once. At the upper or larger end the rolls will take steamboat; a little farther from the end they will take broken; a little farther, egg, and still farther, stove. When the coal to be broken up is of different sizes and the quantity not large, these rolls may be economical.

2861. Saucer Rolls.—Fig. 1029 shows the plan and side view of a set of saucer rolls. There are very few rolls of this type in use, but where they are used they are applied principally to break up the bony coal. As shown in the

figure, they are made up of two saucer-shaped parts, *a* and *b*, which are keyed to the shafts *c* and *d*. To these shafts are also keyed the belt wheels *e* and *f*, which are used to turn *a* and *b* in opposite directions. The parts *a* and *b* are drilled so as to receive the teeth, which are in the form of a knife-blade, made of steel and bolted to *a* and *b*. The part *g*, be-

FIG. 1089.

tween the saucer-shaped parts, is known as the breaking block. It is made of cast iron, and has openings on each side to allow the teeth to pass. The coal is fed on the block *g*, as shown by the arrow *h*, and broken by the teeth as they cover the holes. The coal is delivered, as shown by the arrow *k*, to the hopper located underneath the rolls.

MACHINERY FOR SEPARATING THE SLATE FROM THE COAL.

2862. One of the oldest methods of separating the slate from the coal, and which is still in use, is by hand-picking. The coal and slate are allowed to slide down an inclined chute, along one or both sides of which boys and old men too feeble to perform hard labor are seated; or, in many cases, the slate picker sits astride the chute on a board seat and controls the flow of the coal with his feet. There are several objections to this method. When a large quantity of coal is passing, the slate picker can really pick out only the slate on top, much slate being hidden. Another objection is, one and the same piece of coal having a slaty appearance may be picked up by each slate picker in succession and returned to the chute. For these reasons, different types of picking chutes have been introduced.

As already explained, on account of the way the coal and slate break up, together with the different velocities with which they slide down an inclined chute, numerous automatic slate pickers have been invented. The slate-picker screen, which has already been described, is among the first and the best.

2863. The picker shown in Fig. 1030 is known as the **Houser slate picker,** and consists of a number of separating bars *a*, as shown in the top view *A* and the side elevation *B*. These bars are 4½ feet long, and taper from 1⅛ inches to ¾ of an inch. They are spaced to suit the size of the coal to be handled, and are placed one above the other. The one below is set so as to be in the center of the space of the two above, as shown in the top view *A*. A punched corrugated plate *b* is placed at the upper end of the bars. The holes in it let the fine dirt through. The corrugations are V-shaped and quite deep. At the lower end of the bars are adjustable plates *d* and *c*, so arranged as to catch the coal which drops off the ends of the bars.

The coal to be cleaned is received from the screens upon the inclined plate *e*, from which it slides down over the corrugated plate *b* to the separating bars *a*.

As the coal and slate pass over the corrugated plate the slate is turned up on its edge, is fed into one of the spaces between the bars, and meets with a greater resistance as it moves along than the coal; consequently, the coal jumps farther beyond the end of the bars than the slate does. The

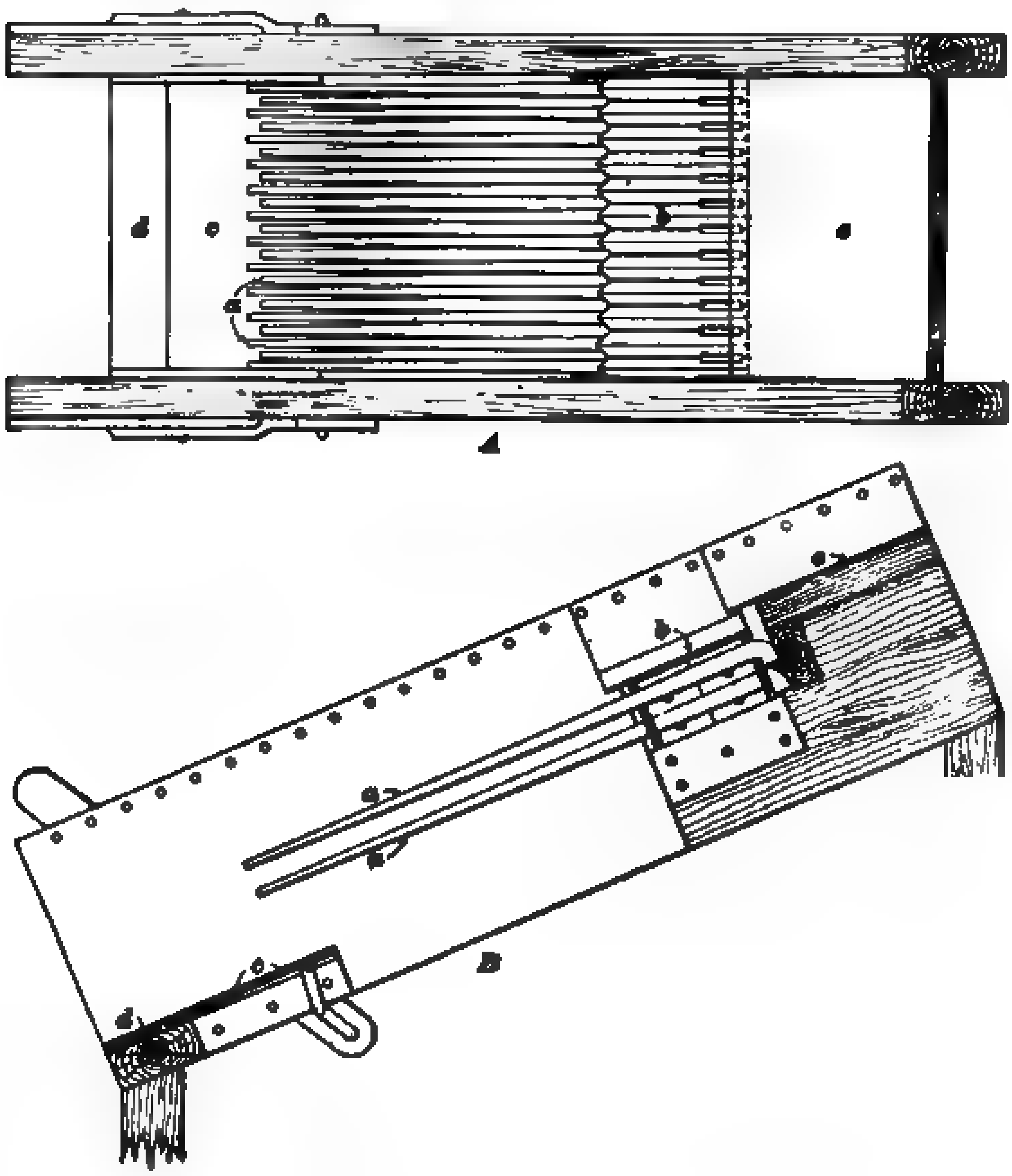


FIG. 1030.

slide *c* is then adjusted to catch the coal and miss the slate which has not dropped through the spaces between the bars.

2864. Fig. 1031 shows a sectional side elevation *A* and a top view *B* of the **Herring separator,** which consists essentially of the feed wheel *a*, the adjustable chute *b*, and

the adjustable plate *c*. The feed wheel *a* is adjustable vertically, that is, it can be raised or lowered, and is driven by a belt which runs on the pulley *d*. The adjustable chute *b* is hinged at *e*, and can receive different degrees of inclination by means of the pin *f*.

As shown in *A*, the chute *b* has the different inclined plates *g*, *h*, *i*, and *j*. The plate *g* is set upon a pitch of 8 inches to the foot; the plate *h* at 5½ inches; the plate *i* at still less, and the plate *j* at 2 inches. The plate *c* is so arranged that

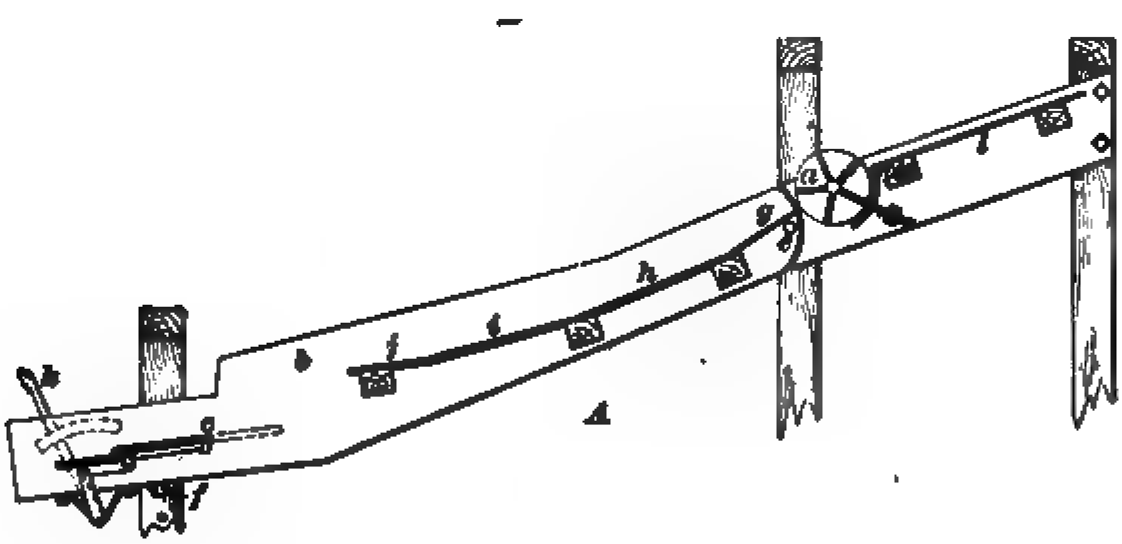


FIG. 1031.

it can be moved back and forth by the lever *k*, so as to adjust the opening for the slate to drop into.

The mixture of coal and slate coming from the screen is received upon the perforated plate *l*, from which it passes to the feed wheel *a*. This wheel delivers the coal and slate in irregular quantities to the inclined chute *b*, down which it slides with a velocity proportional to its frictional resistance, the slate sliding much slower than the coal. The coal acquires sufficient velocity to jump the gap between the

plate *j* and the adjustable plate *c*, while the slate meets with greater resistance and moves with less speed; consequently, the slate falls short of the plate *c*, and drops into the opening which leads to the slate hopper.

2865. Another method very similar to the one just described, and which has been previously referred to, is where the coal and slate are allowed to slide down a chute in which is located a sandstone slab which is rougher than the ordinary chute plate. There is a gap at the end of the slab, over which the coal jumps by virtue of the greater velocity acquired in sliding down the chute, while the slate, moving at a much less velocity, drops into the opening.

2866. Coal Jigs.—All the above methods of separation are used where the coal is comparatively dry. Where the coal is wet and dirty as it comes from the mines, what is known as **jigging** is resorted to.

In jigging, the result is twofold; it washes the coal and separates the slate from the coal.

The separation is due to the difference in the specific gravities of the coal and the slate. The average specific gravity of anthracite coal is 1.473, while the average specific gravity of slate is about 2.5.

The coal and slate are fed into water-tight tanks. Certain laws govern the fall of bodies in water, and in order to apply these laws an arrangement is adopted whereby the particles are repeatedly brought into suspension and allowed to drop again; in other words, a series of blows must be imparted whose magnitude must diminish with the size of the particles. The machine used for this purpose is known as the **jig.**

In the anthracite region two types of coal jigs are in use:

1. The piston jig, with fixed perforated plate bottom.
2. The movable pan, with perforated bottom.

2867. To secure a complete separation of the refuse from the coal in either of the above types of jigs, it is absolutely necessary (1) to have the material to be jigged of uniform size and shape, and (2) to feed the jigs slowly and regularly.

Before being jigged the coal is separated into the various commercial sizes: egg, stove, chestnut, pea, and buckwheat. The sizes above and below these are never jigged.

Screening does not produce a uniform size, for anthracite coal breaks into fragments of almost every conceivable shape, and in any standard size there are pieces that weigh three or four times as much as the smallest pieces. It therefore follows that sizing by screening does not altogether satisfy the first requirement. Again, since much of the slate occurs in flat pieces, they are easily buoyed up by the water and pass over with the coal. Jigging coal in the anthracite region is rarely a cheap process. At a large number of mines the coal must be washed and jigged with mine water that is strongly acid; and the parts of the machinery exposed to the action of this water are rapidly corroded, so that repairs are constantly needed. The cost of jigging coal is, therefore, greatly increased by the necessity of using mine water. In many cases, however, the washed product is more readily salable than the unwashed.

2868. Fig. 1032 shows a side view *A* and plan *B* of a **piston jig.** This jig is a modification of a number of jigs in use, and consists of a water-tight tank divided by the cast-iron partition *a* into two compartments *b* and *c* of unequal size. The tank is lined with tongued and grooved floor boards and the compartments *b* and *c* with iron plates. The bottom of the larger compartment *b* is slightly inclined towards or away from the slate-discharge gate, usually $\frac{3}{4}$ of an inch to the foot, and covered with a perforated plate *d*, supported by the bar *e* in the center, and by supports bolted to the side. The size of the openings in the plate depends upon the size of the coal to be jigged; usually the openings are circular, although for the larger sizes oblong openings are sometimes used, having the longest dimension of the opening in the direction in which the coal moves.

The coal is fed upon the perforated plate *d* at *f*, the inward flow being regulated by the adjustable plate *g*, which is put with its lower end as near the bottom of the jig as is consistent with a free discharge of the coal to be jigged.

FIG. 1002.

The smaller compartment *c*, which is less than half the size of the first, communicates with it across its whole width at the bottom, and serves as a working barrel for a floating piston *h*. To guide the water from the piston to the jig, a semicircular row of planks is put in. These need not be absolutely water-tight, as they are intended simply to direct the current.

The piston *h* consists of a double row of planks placed as shown in the side view *A*, which are either nailed or bolted to the parts *i* and *j*. The lower part *k* of the piston-rod is made of cast iron and bolted to *h*, and is connected to the upper part *l* by means of a 3″ × 4″ oak piece *m*.

The piston receives its motion from cams keyed to the revolving shaft *n*, which is geared to the driving-shaft *o* by means of the elliptical gear *p*.

The object of this gear is to force the piston rapidly down, thereby lifting the coal quickly and allowing it to settle slowly, for the upward motion of the piston is at a much less speed than the downward. In using a single pair of elliptical gears for a quick return, it is well not to have the ratio of the forward motion to the return greater than 1 : 3.

The coal, in rising on the downward motion of the piston, is skimmed off by a series of flat strips of iron carried on two rows of Ewart link-belt chain running over *q*.

As the coal is scraped up the incline *r*, the water drains back. The top *s* of the inclined plane *r*, which inclines slightly towards the jig, is flat and covered with iron. The coal here forms a pile, and the water drains from it back to the jig. As each successive quantity of coal is brought by the flights on the chain, it pushes a corresponding quantity, which has been drained, off the other side *o* down the chute, where it goes either to the picking chute to be picked or directly to the pocket, if it is (as in the case of small sizes) already clean enough. The slate, being heavier than the coal, falls to the bottom and is discharged through the opening *t*. This opening is regulated by elevating or depressing the plate *u*, which is so arranged as to allow the largest pieces of slate to pass under it. The gate *v* generally remains

A

C

Fig. 103

open, but can be closed by the lever attached to the bell-crank *w*. Outside of this is a cast-iron flat pipe, or slate hopper, *x*, which is closed by the wedge-shaped slide *y*, upon the upper surface of which is a piece of oak *z*. The slide *y* moves in a casting *a'*, which has a tapered groove on each side. When the slide *y* is pushed through the opening in *a'*, the wedging action of the taper grooves forces the wood against the face of *a'*. This makes an excellent gate for closing the hopper, allowing neither water nor slate to escape.

The gate *v* is closed by forcing it up in the frame. This cuts off both the water and slate from the hopper *x*. When this has been done *y* is drawn out, the water and slate drop out, and whatever coal may have come with the slate is picked out. The slate is then run either directly to the slate hopper by means of a chute, or conveyed away by an elevator or system of drags.

At *b'* there is an arrangement known as the **slush box,** which lets out the slime and fine coal which accumulate in the bottom of the jig. This is arranged so as not to let out too much water at one time. The gates *c'* and *d'* are similar to the gate *y* on the slate hopper *x*, so that by keeping *c'* closed and opening *d'* a certain quantity of the deposit moves down upon *c'*; *d'* is then closed and *c'* opened, which allows this certain quantity to escape.

The jig pulleys *e'* and *f'* are driven by pulleys from a line shaft in the breaker, and are of such dimensions as to give the proper speed to the piston and the line of scrapers.

2869. Fig. 1033 shows the plan *A*, the side view *B*, and the end view *C* of a jig with a **movable pan having a perforated bottom.** The action of this jig is the same as the one already described, for in every case the object desired is to produce an upward current. The whole pan in this case acts as a piston.

From *A*, *B*, and *C*, it will be noticed that this jig is a water-tight tank. This tank occupies a space 8 feet × 8 feet × 8 feet, equal to an 8-foot cube, and is made of 2½-inch white-pine plank, with braces 5 inches × 6 inches, standards 8

inches × 8 inches, sills 6 inches × 8 inches, top caps 8 inches × 12 inches, all of white pine, making it amply strong and durable. Within the tank is the jig pan *b*. This pan, as shown, is made almost in the form of a complete circle, except at *c*, where the coal passes through the opening in the front of the pan down into the coal elevator *d*. From the side view *B*, it will be noticed that the jig pan *b* is made so as to be slightly inclined towards the coal elevator *d*. This allows the slate that accumulates in the bottom of the pan to move towards the discharge gate *e*, and the coal to overflow at *c*.

The pan in many cases is cast in one piece, but where mine water is used in the jig tank it does not take long for the circular holes to become so enlarged as to make the entire pan worthless. To overcome this, the bottom is made of a number of plates which are bolted to ribs that radiate from the center. These plates, after they are worn out, can be very readily replaced by new ones. From *A* and *B*, it is seen that the pan *b* is made up of two parts.

The sides of the pan proper are cast in one piece with the bottom, having the slate pocket, with gate *e*, bolted to the bottom; the height of the sides is shown by the dotted line in view *B*. The pan proper is surrounded by a heavy piece of sheet iron *f*, which is bolted to the pan as shown in *A*; this is used to prevent the coal from escaping over the rim of the pan.

To keep the pan in position in its up-and-down motion, the shoes *z*, which move up and down on the guides *a*, are bolted to the pan on each side.

The pan, or piston, *b* is suspended in the tank from the line shaft *h* of the small engine *i* at *j*. This engine, which is vertical, has a 6-inch cylinder and 8-inch stroke. The idea of using a separate engine for a jig is to give the person who is operating the jig complete control over it in its separation of slate from the coal, no matter how fast or how slow the coal is passing through the breaker. If the breaker is pressed with a large quantity of coal, the speed of the machinery is naturally slackened, and a jig running with

belting and gearing has its speed also slackened just at a time when the jig should be run fast, to secure perfect separation and do its work properly.

As shown at *j*, the engine shaft is made so as to act as an eccentric. The side view *B* shows how the pole *k* of the pan is coupled to the shaft *h* of the engine *i*. The connecting-rod *l*, which is bolted to the engine shaft, is joined to the pole *k* of the pan by the pin *m*, thus furnishing a hinge joint. In the center of the pan there is a tapering hole, into which the pole *k* is inserted and keyed, as shown at *u*. To the bottom of the pan the slate pocket is bolted, having the slate-discharge gate *e* operated by the bell-crank *v*. The part *w* supporting the bell-crank *v* is bolted to the pan *b*. The lever *x*, in connection with *y*, operates the crank *v*. To the shaft *h* is keyed the gear-wheel *n*, which gears with the wheel *o*, keyed to the shaft *p*. To the shaft *p* is keyed the worm-gear *q*, which in turn gears with the wheel *r* keyed to the shaft *s*. To this shaft *s* are keyed the sprocket-wheels for the coal elevator *d* and the slate elevator *t*.

In many cases friction-gearing is used instead of toothed gearing.

The bottom of the tank is built as shown in the side view *B* and the end view *C*; the bottom is inclined so that the waste coming from the coal will slide down to the slate elevator *t*. The arrangement shown at *g* is used for discharging the tank.

The operation of the jig may be explained thus: The coal and slate to be separated are delivered in the perforated pan *b*, which moves up and down in the tank filled with water at the rate of 180 times a minute.

By this movement the water coming through the holes in the pan creates an upward current, and causes the coal to rise and gradually travel towards *c*, where it passes through the opening in front of the pan down into the coal elevator *d*, by which it is elevated to a chute leading to the coal pocket or to the place where it is to be picked.

The slate being much heavier than the coal sinks to the bottom of the pan, and gradually works its way to the front

of the pan into the slate pocket, and is discharged into the slate elevator *t* through the gate *e*.

FIG. 1034.

2870. Fig. 1034 shows another type of jig with a movable pan; but the pan, instead of being circular, like the one

just described, is rectangular. This type of jig is known as the **Christ patent coal jig.** The figure shows a plan *A* and a side view *B*. This jig has been introduced into the anthracite coal region since the beginning of the year 1895, and at the present time over fifty of them are in operation. The machine is enclosed in a water-tight tank 11 feet long, 5 feet 4 inches wide, and 6 feet 9 inches high. The lining *a* of the tank consists of white-pine plank well corked, so as to prevent leakage. Inside of this lining is another lining *b* of tongued and grooved pine floor boards. The tank has an inclined floor *c* leading to the front and from both sides into a cast-iron boot *d*, placed at the bottom and to the front of the wooden tank.

2871. The principal mechanism of the machine consists of a cast-iron jig box *D*, a longitudinal section of which is

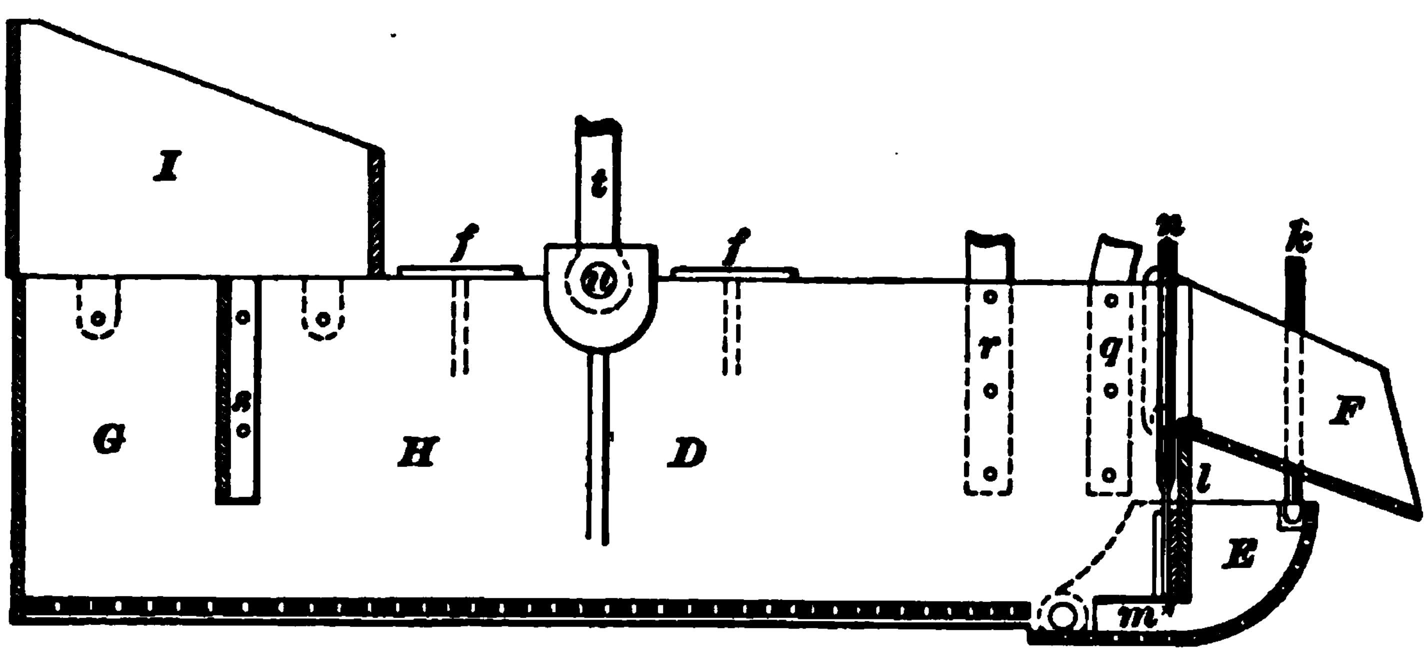


FIG. 1035.

shown in Fig. 1035, which is placed inside of the wooden tank, so that the rear end is at a slight distance from the inner surface of the tank, and the top is about 12 inches from the top of the tank timbers. The jig box is provided on each side with two guides *e* (four in all), which are secured to the box by means of bolts on the extension plates *f* in such a way as to incline the box towards the front end of the tank. Corresponding with these guides are similar ones *g* which rest on plates, and which are in turn fastened to the timbers of the tank so as to make the guides *g* stationary, while the guides *e* are adjusted to move up and down by means of

eccentrics *h* on the main shaft *i*. The shaft *i* is connected with an engine *j*, of about 12 horsepower, fitted with a governor and with the necessary oil-cups.

To the front and at the bottom of the box *D* is a pivoted adjustable gate *E*, which is raised or lowered by means of a hooked screw *k*. The front wall *l* of the box, Fig. 1035, does not extend up as high as the sides, but forms an opening, at the sides of which is pivoted a discharge chute *F* having a perforated bottom.

The main bottom of the box, which does not extend quite to the front, is likewise perforated, as is also the bottom of the pivoted gate *E*. On the inside of the front wall *l* of the box, between the front wall and ribs cast on the side walls, is a secondary plate *m*, which is vertically adjusted by means of the screws *n*, riveted to the plate. The screws *n* and *k* are regulated by means of the shafts *o* and *p*, on which are fitted worms gearing with worm-wheels on the screws *n* and *k*, which are fitted to arms *q* and *r*. At the rear end of the jig box is a cast-iron division-plate *s* riveted to the side walls of the box. This plate does not extend quite to the bottom, but acts as a partition between the two main chambers *G* and *H*. Above the box proper and riveted to its sides is a receiving hopper *I*.

A little to the rear of the center of the box on both sides are sockets, into which extend the eccentric arms *t*, the tops of which are supported by the eccentrics *h* fitted to the main shaft *i*, the bottoms being supported by the pins *u* passing through the sockets.

2872. The Operation of the Machine.—The wooden tank is filled with water to a depth not quite sufficient to cover the jig box entirely when at its highest position, but completely submerging it when at its lowest point.

The engine which revolves the shaft *i* is put in motion, and this in turn operates the eccentrics *h* fastened to the eccentric arms *t*. The eccentric arms give the box a reciprocating motion, which agitates the body of water in the tank. As the box reciprocates, it causes the guides *e* which

are fastened to it to move up and down on their ways *g*. The coal and slate or other material to be jigged is brought by some convenient means to the hopper *I*, and passes down through the receiving chamber *G* to the bottom of the box. The plate *s* prevents the entering material from spreading over the entire surface of the box, as it would otherwise do; it tends to keep the material confined in a small space. As a consequence of this the coal and slate separate, the heavier slate falling to the bottom. All the material then passes through under the plate *s* into the second or jigging chamber *H*.

In this chamber the material forms itself into two distinct layers by virtue of the difference of their specific gravities, the heavier material or slate passing to or remaining at the bottom and the lighter matérial or coal passing to the top. The slate at the bottom forms a layer, the overlapping pieces and the perforations of the bottom plate being to the jig what the valves are to a pump; that is, when the box descends these laps open slightly, permitting the water to pass through and leave the lighter material, and when the box rises, the water is partially prevented from passing down through the material. The material having formed into layers, the next important matter is the limitation and drawing off of these layers. The two layers pass to the front, where the heavier material falls through the opening of the bottom jig box into the pivoted discharging receptacle *E*. This gate, or receptacle, being adjusted by the rods *k*, can be regulated by the shaft *p* while the machine is in motion, so that the adjustment can be of the greatest delicacy. The secondary plate *m* on the inner wall of the box is adjusted in the same way, and is used to increase or diminish the size of the opening through which the slate passes from the jigging receptacle to the slate discharge gate. The plate is useful for accommodating the different sizes of material (that is, one size at a time; for instance, if the jig is working on stove coal, it can be adjusted to take chestnut coal or pea coal, but not at the same time that it is taking stove coal) and also for assisting the gate *E*.

It will be noticed that this gate *E* can be raised to such an extent that there will not be any material whatever passing over it; or it can be lowered to such an extent that all the material passing to the front of the jig will escape through it; or it can be placed at any intermediate point, so that whatever the material to be jigged the height of the front of this gate determines the weight of the refuse that it will take out.

If it is high it will take out only the very heavy material, and if lower it will take out lighter material. The object is to have the front of the gate at just such a point that the slate or heavy material will pass over it and the light material or coal remain in the receptacle *H*, until it rises high enough to pass over the shortened front wall of the box and into the pivoted chute *F*.

2873. An essential point in jigging is that there should be a continuous slate layer at the bottom, and that this should not be varied much. It will be noticed that this is accomplished by having the front of the slate gate always above the bottom of the jig box, thus insuring a constant layer. The top of the slate gate determines or rather marks the line of separation, or that line where the slate layer and the coal layer meet. The coal, after passing into and over the chute *F*, is taken into a second chute at its front, which leads into an elevator boot *v* at the front and about half-way from the top of the tank. The coal elevator *J* takes the coal from this boot and conveys it into a chute which leads to the storage pocket. The slate, after leaving the slate gate, falls onto the inclined floor of the tank, and, together with some coal dirt that passes through the perforations, is taken into the elevator *K* to the front and at the bottom of the tank. The elevator *K* takes it from here to the refuse chute. The elevators *J* and *K* are operated by means of the two sprocket-wheels *w* and *x*, one on the main shaft *i* and the other on the elevator shaft.

It is necessary to slush the tank several times during a day, in order to take the dirty water off and to loosen the

dirt that might accumulate on the floor of the tank. For this purpose a slush gate is provided on the bottom of the slate-elevator boot *d*.

2874. Where coal jigs are used, more especially in jigging chestnut coal, very light slate is buoyed up by the water and carried out with the coal. This slate is removed by the slate picker shown in Fig. 1036, which is placed in the chute leading from the coal elevator.

Where stove and egg coal are jigged, the coal after leaving the jig is hand picked by men and boys.

The slate picker shown in Fig. 1036 is made of iron cast in one piece, and consists essentially of a series of V-troughs, one side *a* of the V being shorter and at right angles to the other side *b*. The lower half of the casting has a taper slit *c* in the short side. The slit is so arranged that anything lying on the long side of the trough and of not too great height can slide out through it. Any lump which is thicker than the height of the slit will, of course, be retained in the trough. The slits widen as they approach the lower end, and the part of the casting below the crossbar *d* hangs freely, so that there is nothing to stop a lump from sliding through the slit. As the coal and slate come down the chute, each lump places itself in one or the other of the grooves or troughs, which are made a little wider than the largest lump of the size for which the slate picker is to be employed. As the lumps slide down, all the flat pieces tend to pass out through the slit on the side, while cubical lumps go over. Should a piece catch in the slit in consequence of the increase in height towards the end, some of the pieces which follow will generally knock it loose, so that it does not remain and block the slits.

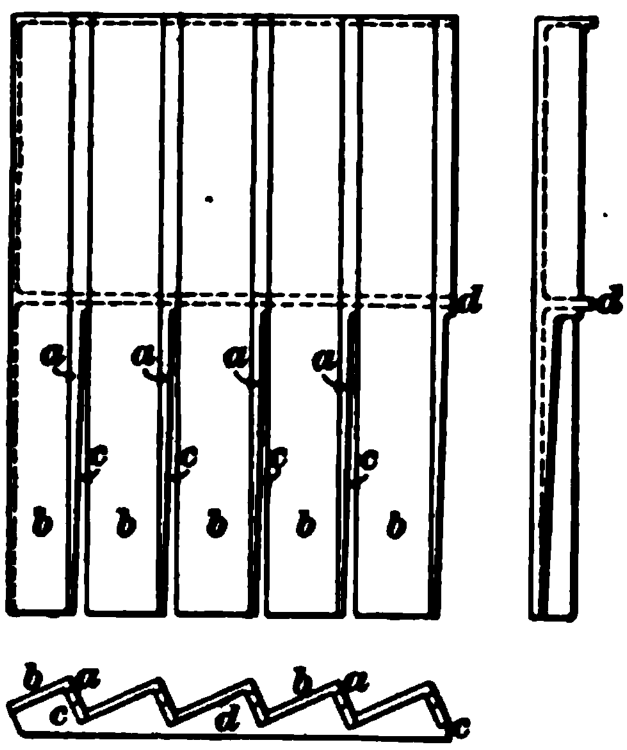


FIG. 1036.

The size and taper of the slit, the pitch of the picker, the

width of the troughs, and the length of the upper and lower portion of the casting vary with the size of the coal and the nature of the slate. This class of slate picker is sometimes used for the larger coal, such as broken and egg. It is also used where there is a great amount of flat coal, which, while unsightly and unmarketable in its flat shape, is nevertheless pure coal. In this case the flat slate and coal coming from this picker is run to a separate set of rolls and broken into smaller sizes.

MACHINERY FOR CONVEYING THE COAL IN THE BREAKER.

2875. This is a class of machinery that is generally used for changing the position of the coal, although, as was shown in Fig. 1024, a part of it is also used in the preparation of the coal. It comprises chutes, elevators, drags, or conveyors, and loading lips.

2876. The **chutes** are used for conveying the coal and slate from one part of the breaker to the other, and are inclined downwards, upwards, or are horizontal. Those that are inclined downwards convey the coal from a higher to a lower level, the coal and slate sliding down by gravity. As a rule, the cross-section of this class of chutes is in the form of a rectangular trough, the depth and width depending upon the size of the coal or slate and the amount that is to be carried down the chute. The pitch depends upon the size and quality of the material.

Fig. 1037 shows the cross-section of a chute as ordinarily constructed. The bottom consists of a row of planks *a* spiked to the support *b*. The sides *c* are spiked to the bottom plank *a*, and are covered, as a rule, with either cast plate or sheet iron *d*. The side iron *d* is put on before the bottom iron *e*, so that the bottom iron will fit between the side irons, for it is necessary to change *e* oftener than *d*.

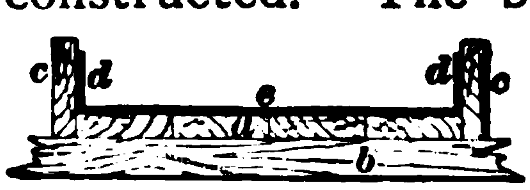


FIG. 1037.

The thickness of the iron depends upon the work to be

performed; the larger the coal the heavier the iron, and the smaller the coal the lighter the iron and the greater the pitch of the chute.

2877. The following table contains the pitches per foot of the chutes ordinarily used in the breaker, down which different sizes of coal will slide.

TABLE 50.

Size of Coal.	Pitch in Inches per Foot.	
	Dry Coal.	Wet Coal.
Rice	9	2½
Buckwheat	7	4
Pea	6½	3½
Chestnut	5	3½
Stove	4½	2
Egg	4 (shelly coal, 4½)	2
Broken	4	
Steamboat	4	
Lump	4	

From the above table it will be noticed that the inclination for rice coal, when it is prepared wet, is very slight. Where it is prepared wet the chute is generally lined with terra-cotta pipe (cut in half), and a considerable quantity of water allowed to run down the chute, so as to carry the coal to the pocket.

2878. Some chutes, instead of having a rectangular cross-section, are half-round troughs; and where an abrupt change in the direction of the chute becomes necessary, cast-iron turns are often used, which are spiral, half-round troughs of greater or less length.

Fig. 1038 shows a cross-section that is frequently used when the chutes are inclined upwards or are horizontal. It is a cast-iron trough nailed or secured to the side supports, which are of wood, and is used for a system of drags where the coal or slate is to be elevated from a lower to a higher level, or conveyed from one point to another on the same level.

FIG. 1038.

2879. Elevators.—Elevators are used for raising coal or refuse in the breaker. The kind shown in Fig. 1039, known as the **link-and-bar elevator,** is used where the coal is to be elevated to a considerable height.

In the figure *A* shows the side elevation and *B* the end view. The elevator consists essentially of the bucket *a*, the link *b*, the bar *c*, and the wedge *d*, the details of which are shown in *C*. The bucket *a* is made in two parts, the front *f* and the back part *g* being riveted together at the sides. The front *f* contains a double set of holes, which are used for the inside links *h* and the outside links *k*, as shown in *B*. These buckets are usually made of sheet iron in different sizes, 14 inches × 18 inches being a very common size.

The link *b* is made of strap iron, bent to the proper shape and welded together, after which the two bolt-holes are drilled. For a 14-inch × 18-inch elevator-bucket, the links are made of 2-inch × ⅝-inch iron.

The bar *c* is made of wrought iron and has the two collars *l* and *m* welded on, so as to keep the links *b* in the proper position. The bars are made to extend from 3 to 3½ inches over the outside of the buckets, and are run in wooden guides, not shown in the drawing. For a 14-inch by 18-inch elevator-bucket, the bar is of 1¼-inch iron.

The wedge *d* is single-tapered and made of oak, having two holes bored in it to permit the bolts which fasten the buckets to pass through it. The wedges are used to make the buckets stand out from each other: for if the buckets were bolted

directly to the link it would be impossible for them to pass over the top and under the bottom wheels, on account of the parts not being flexible enough.

The wheels *n* and *o* at the top and bottom, over and under which the buckets travel, are known as the spiders. They are made of cast iron, bored and key-seated, so that they

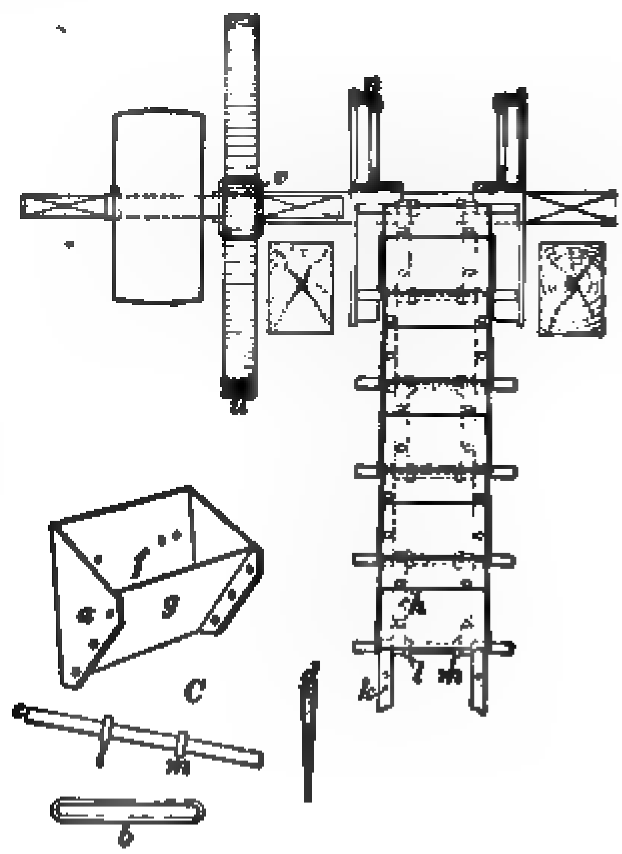


FIG. 1089.

B

can be keyed to the shafts *p* and *q*. The shaft *p* must be of much greater dimensions than the shaft *q*, for it carries the load, while the shaft *q* merely keeps the spiders that it is keyed to in place.

The bottom stands *r* have sliding boxes *s*, and by means of the set screws *t* the wheel *o* can be raised or lowered, thus

taking up the slack that may be found to exist. The spur-wheel *u*, as generally used, is very large and the pinion *v* small, for this class of elevator is never run rapidly. As a rule, for this class of elevators (with 14-inch by 18-inch buckets and upwards) a spur-wheel with 4-inch face is used where the shafts are 35 feet apart and less; if over 35 feet a 5-inch face is used.

2880. In many places the same style of bucket is in use as shown in Fig. 1039, bolted to a heavy rubber belt, running vertically or inclined. There are other styles of elevators used where the height is not so great as it is in cases where the link-and-bar elevator is used.

Fig. 1040.

2881. Fig. 1040 shows a **traction-wheel elevator,** with detachable link belting and cast-iron boxes.

Fig. 1041 shows another type of elevator, known as the **double-chain elevator,** fitted with a cast-iron boot, the boot being fitted with a take-up.

2882. Drags, or Conveyors.—In recent years the methods of handling all kinds of materials have received the attention of the engineering profession in a marked degree, and the result is that large numbers of mechanical conveyors are successfully used in all parts of the country. Conveyors,

Fig. 1041.

drags, or scrapers, as they are often called, are a class of machinery that has found much favor in the anthracite region, and may be used in either horizontal or inclined chutes. Most of the drags or conveyor lines in use in the anthracite region are constructed with the Dodge conveyor chain or the Ewart chain. This class of machinery is used principally for conveying coal, slate, and culm in and about the breaker where it is found impossible to put a chute, and where it is found more convenient and cheaper to use drags than to use a small dump-car or wheelbarrow.

2883. Fig. 1042 shows the elements of conveyors as commonly used in anthracite-coal breakers.

FIG. 1042.

In this figure, *A* is known as the **flight conveyor,** *B* as the **upper and under run conveyor,** and *C* as the **drop-flight conveyor.**

These drags, as shown, are made up of a trough or chute, at each end of which are sprocket-wheels, over which the link chain carrying the conveyor flights is run. These flights are fastened to the chain by a special link inserted in the chain. The drags are also fitted with a take-up, so as to take up any slack that may occur. At the driving end of the line the driving-wheel is either keyed to the same shaft as the sprocket-wheel, or a pinion and spur-wheel is used.

These drags are driven by a rubber belt, by a wire or hemp rope, or by link belting, as shown in Fig. 1043. The

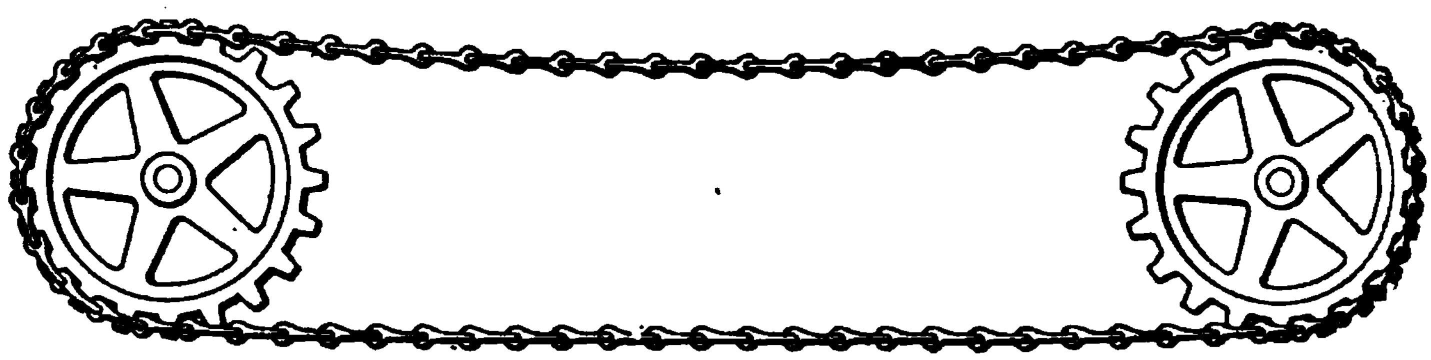


FIG. 1043.

principal value of link belting as a power transmitter lies in the nature of its construction; since, being composed of links and used with sprocket-wheels, it forms a positive belt and prevents any loss of motion through slipping, as is often the case with rubber belts or hemp ropes. Where the power is to be transmitted at a slow rate, the link belting is far superior to anything else.

2884. Loading Lips.—These are specially designed chutes to convey the coal from the pockets located in the bottom of the breaker to the bottom of the railroad-cars, so as to prevent breakage. There are two types of loading lips, one for lump and steamboat coal and the other for the smaller sizes.

2885. Fig. 1044 shows a side view of a loading lip for lump and steamboat coal. It consists essentially of the apron *a*, made of sheet iron and hinged to the main chute at *b*. The car to be loaded runs under the chute, and the apron is let down into the car by means of the lever *c*. The gate *e* is then opened, and the coal is allowed to run over

the lip screen *f*, which takes out the finer coals. By raising or lowering the apron and regulating the flow of the coal at

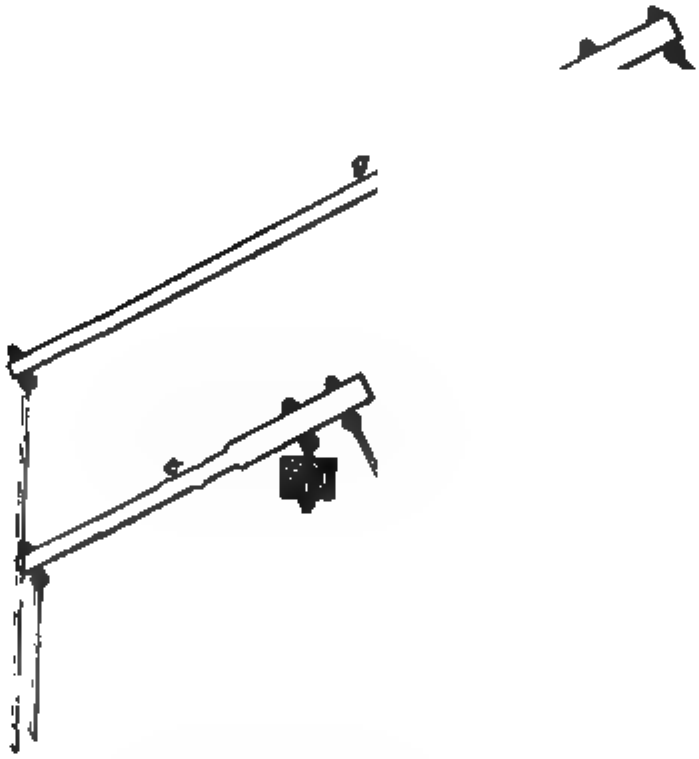


FIG. 1044.

the gate by means of the lever *g*, the coal can be deposited in the car as desired.

FIG. 1045.

2886. Fig. 1045 shows a sectional view of a loading lip for coal below the size of lump and steamboat; it is known as the **Griffith loading chute.** This apparatus consists essentially of a rectangular wrought-iron trough *a*, curved to part of a circle and resting on guide-rollers *b*, *c*, and *d*. It has a hood *e* and a gate *f* on its front end, the bottom extending back to *g*, a few inches under the pocket chute containing the lip screen *h*. It is cut away here, so that the operation of the screen *h* and the small chute *i* will not be interfered with while the loading chute is in use, nor when it is drawn back out of the way of cars.

The weight of the chute is nearly balanced by the chain *j* and weights *k* and *l*; the remaining part of the weight is carried by the hand chain *m* and the weight *n*.

The chute runs forward as soon as the chain *m* is slackened off. The small chain *o* is for manipulating the gate *f*. The waste coming from the lip screen *h* is conveyed by the chute *p* to the conveyor line, which runs in the trough *q*.

DESCRIPTION OF AN ANTHRACITE BREAKER.

THE CONSTRUCTION.

2887. Figures 1046, 1047, and 1048 show the plan, elevation, and cross-section, respectively, of an anthracite breaker.

This breaker, as shown, is not arranged for any particular opening, but can be used for shaft, slope, drift, tunnel, or stripping. It is so arranged that part of the coal is prepared wet and part dry.

The general plan of the structure is a wooden trestle.

2888. The Plan View.—The plan, Fig. 1046, shows the arrangement of the posts and the general arrangement of the machinery used in the breaker. It also shows a diagram of the process of the preparation of anthracite coal for market.

vs the arrangement of the screens and other ma-ery.

:om the elevation it is seen that the breaker is located sloping surface. The piers *A* are arranged in steps, the continuous walls *B* are almost all on the same This figure also shows the method of framing and ng. Instead of using the ordinary mortise and tenon aming, cast-iron brackets m_6 and m_7 are used for many e larger sized timbers. As shown, these cast-iron ets m_6 and m_7 have two small projecting arms m_8 and hich are set into the mortised timbers. Generally, racket has six holes drilled in it to receive the bolts—r the upright posts and two for the cross-timbers—one bolts doing for two brackets, as here shown, the bolt-in the brackets being set in line with each other.

transverse timbers, as shown by m_{10}, are also secured ackets. As shown in the figure, these timbers are a above the timbers shown in the side view, so that the will not meet each other. The braces are nearly all or 30° to the uprights and secured by oak pins.

92. In this figure n_3 shows a mine-car in the position umping, or being tipped; the part n_3, covered by the n_4 and having the floor on which are spiked the rails known as the tip house, or the place where the mine-are dumped. The dump n_6 is known as the cradle p, which has already been spoken of.

he coal as it comes out of the mine-car passes over the s n_7 to n_{11}, which are generally spoken of as the main eening or platform bars; they are set 3½ to 5 inches rt.

The coal that drops through the bars n_7 to n_{10} passes into e V-shaped hopper o_3. This hopper is made the entire dth of the screening bars, and in many cases it is referred as the **main hopper.** As shown, the bottom o_4 of this opper rests on very heavy beams o_5, which are notched into each other at o_6. The bottom is always very heavily ironed, but the sides are lined with a very much lighter iron. There are three openings at the lower end to feed the coal out of

the hopper o_3 into the circular hoppers o_7, which convey the coal to the three mud screens l, l_1, and l_2, l being the only one shown, as the other two are parallel to it.

The three mud screens l, l_1, and l_2 are built very short, and are used to obtain only a partial separation of the coal. They are single-jacketed screens having three rows of segments, and are constructed in the same manner as those already described. From this view, it is seen how the small pinion underneath the screen meshes with the screen gear. It will be noticed that the pinion is keyed to a shaft that is horizontal, while the screen shaft to which the large gear is keyed is set at an angle, so as to give the required pitch to the screen. The back ends of these screens are supported by hangers, which are bolted to the cross-timbers o_8. The journals at the front end of the screens are placed in specially designed boxes that rest on the cross-timber o_9.

The side elevation shows how these screens are driven; for from the plan it is evident that the mud screens l, l_1, and l_2 are driven by the pulleys m and m_1, m being keyed to the main line shaft V, and m_1 being keyed to the shaft m_2. From this view, only one of the three miter gears can be seen, m_3, which is keyed to the shaft m_2.

Directly above these three mud screens are located the three water troughs p_3, p_4, and p_5. In this view the trough p_3 obstructs the view of p_4 and p_5, which are arranged parallel to p_3. These troughs are kept overflowing, the water being fed by a pipe p_6 in the bottom of the troughs. The feed-pipe p_6 connects with the pipe p_7, which is connected to the water-tank p_8. The water is pumped into this tank through the pipe p_9. In many cases the pump is located in the interior of the mine. The tank p_8 is located above the troughs p_3, p_4, and p_5, so as to get enough head to keep the water overflowing in the troughs.

The coal that drops through the bars n_{11}, which are 5 inches apart, passes into the main or No. 1 rolls, which are marked C.

2893. The inclined floor q_6 is covered with cast plates on a pitch of $1\frac{1}{2}$ inches to the foot, and is known as the

platform. The coal that has passed over the main screening or platform bars is cleaned and assorted on this platform q_6 by a number of men known as platform men. This platform consists of symmetrical halves; that is, it has two holes to take the coal to the main rolls C, and two places q_7 to throw the chippers to be chipped. The inclined part q_8 is where the lump coal and heavy rock are pushed over by the platform men into the lump-coal and rock chute q_9, which is one chute, but divided by a partition, the lump coal being allowed to run on one side and the rock on the other.

The beams q_{10}–q_{14}, shown in this figure, are used to support the lump-coal and rock chute, the planking for the floor of the chute being removed.

Another view of this chute is shown by q_{15} q_{16}; the lump coal and rock coming over the top of the breaker change their course by dropping into the chute q_{15} q_{16}, located at right angles to the chute supported by the beams q_{10}–q_{14}. The lump coal, in running over q_{15} q_{16}, is allowed to drop through an opening in the bottom at q_{17} into another section of the chute q_{18} q_{18}, which is directly under q_{15} q_{16}. The rock which runs parallel to the lump coal passes over the end of q_{15} q_{16} into the slate chute q_{20} q_{20}, which is located directly underneath the lump-coal chute q_{18} q_{18}. The rock from chute q_{20} q_{20} is loaded into dump-cars that run over the rails l_{10} and l_{11} to the rock bank.

Underneath the lump-coal chute q_{18} q_{18} is located the V-shaped pocket q_5 used to hold the lump-coal screenings, which are the smaller pieces of coal that have been broken off the larger lumps that drop through the bars located directly above the pocket. These lump-coal screenings are loaded into a small dump-car that runs over the rails l_7 and l_8, and conveys them to the lump-coal screenings elevator d_{12}, whence they are lifted into the breaker to be reseparated.

The hopper r_6 is for the main rolls C; it is circular in section, and is used to convey the coal from the rolls after it has been crushed. This hopper, as shown, is on a good pitch, so that there is no danger of the coal stopping up and thus "stalling" the rolls.

2894. The coal, after leaving the hopper r_8, drops into r_6, where it is divided, the one part going to the steamboat screen a_{11} and the other part to the steamboat screen a_{12}.

The steamboat screens a_{11} and a_{12} are both of the same dimensions, and are arranged parallel to each other. They are double-jacketed cast-iron screens with four rows or sets of segments, steamboat-coal coming out of the end of the inside jacket and broken coal coming out of the end of the outside jacket.

As shown, the steamboat screen a_{11} is driven by the screen gear a_9, which gears with the small pinion a_7. From this view it is shown how the rope pulley, a_3 and a_5 are located, a_3 being driven by the rope pulley a_1, which is keyed to the main line shaft V. The driving-gear for the other steamboat screen a_{12} is arranged parallel to the driving-gear of the screen a_{11}, and all the parts of it are of the same dimensions as those shown in this figure. The back ends of the screens are supported by hangers which are bolted to the beam r_7. The journals at the front end work in specially designed boxes which are bolted to the beam r_8.

The chutes r_9 and r_{10} carry the steamboat and broken coal that comes from the steamboat screens a_{11} and a_{12}. Parallel to the chute r_9 is the chute r_{11}, which carries the coal that comes out of the front end of the three mud screens l, l_1, and l_2. The coal that travels down chutes r_9, r_{10}, and r_{11} is picked in the picking room r_{12}.

The steps r_{13}–r_{19} show the arrangement of the floor in the picking room r_{12}, which consists of a series of steps, or short platforms, arranged one above the other.

2895. The hopper s_4 for the mud screens is cut away to show the partition s_5, which is used to separate the fine coal from the coarse coal. The chute s_6 carries the coarse coal that drops out of the two rows of segments on the three mud screens. This coal is conveyed to the wet egg-coal screens b_{11} and b_{12}. The chute s_7 carries the fine coal that drops out of the segment on the back end of these three mud screens. This coal is conveyed to the two wet pea-coal

screens y_5 and y_6. The chutes s_6 and s_7 are made water-tight to prevent the water from constantly dripping.

The number of circular rings that compose the figures b_{11} and b_{12} are the projections of the circular screen rings to which the segments are bolted that make up the wet egg-coal screens. The inside circles are used to support the inside jacket, and the outer circles to support the outside jacket.

The small circle marked b_8 is the pinion that meshes with the screen gear b_7, which in turn gears with b_9. The circle marked b_2 represents the belt wheel which is used in the driving of the wet egg-coal screens b_{11} and b_{12}. The circle marked k_1 represents the belt wheel which is keyed to the driving-shaft for the wet egg-coal screen, and is used to drive the lip-screenings elevator k_{10}.

The number of circular rings that compose figures y_5 and y_6 are the projections of the circular screen rings to which the segments that go to make up the wet pea-coal screens are bolted.

The small circle y_2 represents the pinion that meshes with the screen gear y_3, which in turn meshes with y_4. In this case the pinion, instead of being underneath the screen, is on the top. The circle y_1 represents the belt wheel that drives the pea-coal screens.

2896. The screen q_5 shows the arrangement of a slate-picker screen which is arranged parallel to the slate-picker screen q_4, as shown in the plan. These screens are made up of three sets of cast-iron segments, known as **slate pickers.** As shown, the longitudinal openings in these pickers through which the flat slate drops out are at right angles to the screen shaft.

There are three cast-iron rollers on the top of each screen, which are not shown here, one for each set of slate pickers. These are used to keep the openings in the slate pickers from becoming clogged, for if they were not used, the flat slate would soon block up the openings. The slate screen q_4 is used for taking slate out of the stove coal as it comes

from the meshes of the stove-coal segments on the wet egg-coal screens b_{11} and b_{12} before entering the stove-coal jig. The slate screen q_8 is used for taking slate out of the chestnut coal as it comes from the meshes of the chestnut-coal segments on the wet egg-coal screens b_{11} and b_{12} before entering the chestnut-coal jig. These screens are driven by the bevel-gears, as shown by q_3, located at the front end of the screens, which in turn are driven by the belt pulleys q_1 and q, q being keyed to the main line shaft V.

2897. The tank w shows the end view of the buckwheat-coal jig tank. The tanks for stove, chestnut, and pea coal jigs are arranged parallel to the tank w. The chute t_2 shows the inclined drag-line chute for buckwheat; t_3, shown in the plan, is for pea coal, and is arranged parallel to t_2; they convey the coal from the jig tank w and w_1 to chutes which convey the coal to the buckwheat and pea coal pockets.

The chute t_4 shows the inclined drag-line chutes that convey the coal from the stove-coal jig tanks to the picking chutes, where it is picked before entering the pockets. The chutes t_5, shown in the plan, are arranged parallel to the chutes t_4, and are used for conveying the chestnut coal from the chestnut jig tanks to a chute which leads to the chestnut-coal pocket.

The belt wheel v_1 is in line with the belt wheels s_1 and r_1, and shows the side view of the three belt wheels which are arranged parallel to each other. They run the line shafts for the jig pistons of the buckwheat and pea and stove and chestnut coal jigs. One of the cams u_4 which gives the up-and-down motion to the jig pistons is also shown in this view.

2898. The circle h_1 shows the side view of the rope pulley that is used in connection with the driving-gear of the main elevator u_5.

This elevator is the regular link and bar elevator, a detail of which is shown in Fig. 1039. In Fig. 1039 the small pinion is geared at the side instead of directly under the large spur-wheel, as shown in this view. The elevator u_5 is

used to elevate all the dry coal that is cracked up by the Nos. 2, 3, 4, and 5 rolls, also that coming from the lip screens under the breaker.

The screen k shows a side view of the broken-coal screen, which is a single-jacketed screen having three rows of segments. As shown, the screen gear meshes with the small pinion above the screen. The shaft to which this pinion is keyed runs directly over the center of the screen, and at the front end of the screen is the miter gear j, which is driven by the rope pulleys i_1 and i, i being keyed to the shaft of the breaker engine. (See also the cross-section, Fig. 1048.) The pulley i_1 is keyed to the same shaft as g_2, which is shown in this side view. The pulley g_2, in connection with the pulley g_3 (shown in the plan), drives the drag-line g_5, which conveys the broken coal that comes out the end of the broken-coal screen k, after it has passed down a short chute leading to the drags, to the picking chutes, where it is picked before entering the pocket.

The hopper u_6 carries to the main elevator u_5 (see also the plan and cross-section) the different sizes below broken coal that drop through the meshes in the broken-coal screen.

2899. At different times there is a greater or less demand for one size of coal than for any other. When there is little demand, or no demand at all, for broken coal, instead of hoisting it up in the drag-line g_5, it is run into the No. 3 rolls E and broken up into smaller sizes; in case there is no great demand for egg coal at the same time that there is no demand for broken, by jacketing the broken-coal screen with segments of a smaller mesh, the egg and the broken coal will come out the end of the screen.

2900. The hopper u shows the side view of the roller hopper for the No. 3 rolls E, which is used to carry the coal, after being crushed, away from the rolls to the main elevator u_5.

The bony coal coming from the broken, egg, and stove coal pickings is broken into sizes below stove coal by the No. 4 rolls F.

All the flat slate and coal coming from the slate-picker screens q_4, q_5, d_7, and d_8 are broken into sizes below chestnut coal by the No. 5 rolls *G*.

The coal coming from the No. 4 and No. 5 rolls is conveyed by the roller hopper u_6, a side view of which is shown in this figure, to the main elevator u_5.

2901. The inclined timbers v_4, v_5, v_6, and v_7 are the supports for the waste pocket *W*. This pocket extends almost the entire width of the breaker, and collects all the waste, such as culm, rock, and slate. The culm, however, coming from the wet rice-coal screen, is carried off in troughs with water and deposited on the slush bank.

The inclined timbers v_4 to v_7 are covered, as shown, with 3-inch planks, which form the bottom of the waste pocket. These planks are covered with cast-iron plates or very heavy sheet iron.

In this pocket, v_8 shows the very fine culm coming from the dry rice-coal screen; v_9 is the slate coming from the picking rooms, and v_{10} shows the slate coming from the buckwheat, pea, chestnut, and stove coal jigs.

At the bottom of this waste pocket *W*, on both sides of the tracks, a number of loading gates are arranged to load the waste into dump-cars that run over the double tracks formed by the three rails l_9, l_{10}, and l_{11}. The position of the No. 2 rolls is shown at *D*. These are commonly spoken of as the prepared coal, or monkey rolls. They break up all the coal coming from the mud screens *l*, l_1, and l_2, and at times the coal coming from the steamboat screens a_{11} and a_{12}.

2902. The hopper w_3 shows a side view of the circular hopper for the No. 2 rolls *D*, which conveys the crushed coal coming from the rolls to the broken-coal screen *k*.

The screen x_4 shows a side elevation of the wet rice-coal screen. There is another rice-coal screen arranged parallel to this, and of the same dimensions, which is used to prepare the rice coal on the dry side. Both are single-jacketed screens having four rows of segments.

As shown, the wet rice screen x_4 is driven by the large

screen gear x_4, which meshes with the small pinion x_3 over the screen. This pinion is keyed to the same shaft as the rope pulley x_1, which is driven by the rope pulley x keyed to the main line shaft V. The circle x_2 shows the position of two wheels, called deflecting wheels or pulleys, which are used to change the direction of the rope that drives the pulley x_1. These wheels work loosely on the same axle, that they may turn in opposite directions. (See extreme right of end elevation.)

The screen d_7 is a cast-iron slate-picker screen, and the plan shows another similar screen d_8. These screens d_7 and d_8 are of the same dimensions and are arranged parallel to each other. Both of these screens are of the same dimensions as the slate-picker screens q_4 and q_5, and have the same arrangement of machinery in use above the screen for their operation. The screen d_7 is used for taking the slate out of the chestnut coal as it comes from the meshes of the chestnut-coal segments of the dry egg-coal screens b_{13} and b_{14}, as shown in the plan; similarly, the slate screen d_8 is used for taking the slate out of stove coal as it comes from the meshes of the stove-coal segments on the dry egg-coal screens b_{13} and b_{14}.

2903. To the shaft d_4 is keyed the bevel-gear wheel d_6, which is used in driving the slate screen d_7. To this same shaft d_4 are keyed the pulleys d_9 and d_3, which are of the same dimensions; d_9 obstructs the view of d_3.

The pulley d_9, in connection with the belt pulley d_{10} and the gearing d_{11}, drives the lump-coal screenings elevator d_{12}, all of which are shown in this side elevation. The rope pulley d_3, which is keyed to the shaft d_4, is used in connection with the rope pulley d_2, keyed to the main line shaft V, to drive the screens d_7 and d_8 and the lump-coal screenings elevator d_{12}.

The coal that is elevated by the lump-coal screenings elevator d_{12} is conducted by a chute to the broken-coal screen.

The circle u_3 shows the position of the two deflecting rope

pulleys which work loosely on the shaft. They are used in connection with the rope pulley u_1 and the gearing u_2 in driving the stove-coal drag-line t_4. These pulleys u_3 are put in to keep the driving rope above the pea-coal screens.

The circle z_4 also shows the position of deflecting pulleys which work loosely on the shaft. They are used in connection with the rope pulley z_1 and the gearing z_2 in driving the buckwheat and pea coal drag-line t_2. The rope pulley z_3 is put in to keep the driving rope away from the side of the wet rice-coal screen.

2904. The chute w_3 shows the side view of the picking chutes on the wet side of the breaker, where the coal as it leaves the drags and screens is picked by men and boys as it passes down the picking chutes on its way to the pockets.

From w to w_6 is shown the arrangement of the floor in the main picking room.

The coal, after leaving the picking chutes w_3, passes along chutes w_7, which are known as *telegraphs*. These are chutes with no regular sides. The iron that forms the bottom is slightly turned up on each side, so that the coal can leave the chute from the sides as it blocks up from below. They are used for conveying coal or slate to the pockets. In this figure, the arrows placed along the sides of the telegraph w_7 show that the coal, after filling the pocket w_8 from below up, can drop off on either side, and thus completely fill the pocket.

2905. The bottoms of the V-shaped coal pockets w_8, the end view of which is shown in this figure, are supported on beams w_9 and w_{10}, which are on a pitch of 8 inches to the foot. These beams are well supported by the posts and braces forming the bents *1*, *2*, *3*, and *4*. The coal from these pockets can be loaded from either of the loading chutes w_{11} or w_{12}. The coal that is loaded from the chute w_{11} is loaded into house-cars or large gondola cars, which run on the track the cross-section of which is shown in l_3 and l_4. The coal loaded from the chute w_{12} is loaded into the ordinary railroad-cars that run on the track formed by the rails l_5 and l_6.

At x_8 and x_7, in the loading chutes w_{11} and w_{12}, are placed the loading lip screens which take out the fine stuff, which drops into the chute x_8, and is conveyed by the drag flights running over the sprocket-wheel k_{21} to the lip-screenings elevator k_{10}.

The rope pulley k_{12}, in connection with the miter gear, as shown in the plan, and the bevel-gearing k_{16}, as shown in this side elevation, is used to drive the drag-line which passes over the sprocket-wheel k_{21}.

The rope pulley k_{12} is on line with the rope pulley k_{11}, which is keyed to the shaft k_7; to the shaft k_7 is also keyed the rope pulley k_2, which is driven by the rope pulley k_1, keyed to the shaft that drives the egg-coal screens, by means of the idler k_3, which consists of two deflecting pulleys, used to change the direction of the driving rope that runs over the rope pulleys k_1 and k_2.

The screen k_6 shows the side view of the lip-screenings separator, which is in part a double-jacketed screen, driven by the small pinion k_4, which meshes with the screen gear k_5.

The pinion k_4, as shown, is keyed to the shaft k_7, to which is also keyed the pinion k_8, gearing with the spur-wheel k_9 of the lip-screenings elevator k_{10}, which empties its contents into the screen k_6. This screen k_6 takes out the buckwheat and pea coal, the buckwheat coming out of the end of the outside jacket, and the pea coal passing through the meshes of the single-jacketed portion of the screen. The buckwheat is carried by the chute y_7, and the pea coal by the chute y_8, to the pockets located on each side of the screen. The fine stuff that drops through the meshes of the outside jacket, together with the coarse coal coming out of the front end of the screen, is carried through the hopper y_9, which is only partly shown, to the main elevator u_8.

The breaker engine a is located in the lower part of the breaker, and is protected from the dust, etc., by the roof z_4.

2903. The roofs that cover the different parts of the structure are shown in this view. They are supported by much lighter posts than those used in the lower part of the

breaker structure, and are mortised into transverse beams. The post caps are of the same dimensions as the posts, and support the rafters, which are 4 inches × 6 inches in section, set on end, and the whole covered with No. 26 corrugated iron sheets, as shown in Fig. 1050. The sheets have a lap of at least two corrugations on the sides, and from 4 inches to 6 inches top and bottom. Where the posts are very far apart, the rafters, instead of resting directly on the caps, are supported by corbel blocks, as shown in the figure.

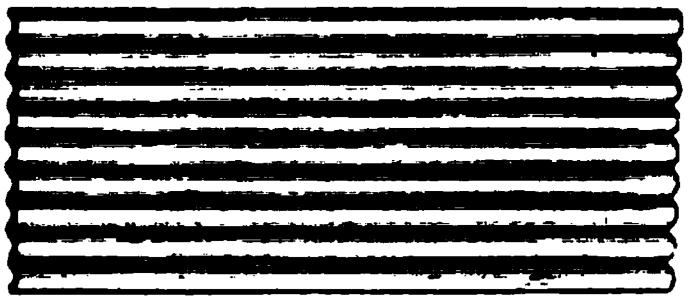


FIG. 1050.

This corrugated iron is manufactured in sheets 6, 7, 8, 9, and 10 feet long, or, if desired, is furnished in special lengths, and is coated with a red mineral paint. The painting is all done by machinery, which is much better than with the hand brush, as it insures an even coating. The sheathing for the different sides is also formed of these corrugated sheets.

Between the upright posts, 3-inch × 4-inch horizontal bracing timbers are placed, which are used as nailers. These are spaced so as to suit the lengths of the sheets.

2907. The Cross-Section.—Fig. 1048 shows the cross-section. Since most of the main features have been previously described, only those which are more clearly shown by this view will be mentioned here. The student should constantly refer to this view when reading the description of the side elevation.

From this cross-section it can be better understood how the large chippers are taken off the platform q_6, and thrown to the chipping platforms q_7, where they are chipped. The lines ——-—— marked a_{14} and a_{15} represent the chutes leading from the chipping platforms q_7 and the openings in the main platform q_6 to the main rolls C, where the pure coal is thrown after the slate has been separated, the rock and slate being deposited in the rock chute.

The circular rings composing figures l, l_1, and l_2 are the

projections of the circular screen rings that go to make up the three mud screens l, l_1, and l_2.

C shows a side view of the main rolls with the belt pulley d_1, which is in line with the belt pulley d, keyed to the shaft of the breaker engine.

In this view the line ——-—— marked r_8 shows the roller hopper through which the coal is carried away from under the main rolls C.

The line ——-—— marked r_9, which branches out from r_8, shows the hoppers leading from the roller hopper r_8 to the steamboat screens a_{11} and a_{12}. These screens are shown in this view by the number of circular rings which are the projections of the circular screen rings that make up the screens.

2908. This cross-section will explain how the hopper s_4 or the mud screens l, l_1, and l_2 conducts the particles that drop through the meshes of these screens to the chutes s_6 and s_7, Fig. 1047. These chutes carry the coal until they meet the chutes b_{15} and b_{16}, which are at right angles to the chutes s_6 and s_7. The chute b_{15} carries the coarse coal into the wet egg-coal screens b_{11} and b_{12} on the wet side, while the chute b_{16} carries the fine coal coming from the back end of the mud screens l, l_1, and l_2 to the hopper f_9, under the egg-coal screens b_{11} and b_{12}, through which it reaches the pea-coal screens y_8 and y_9 on the wet side.

The hopper b_{17}, under the steamboat screen a_{11}, catches the coal dropping through the meshes of this screen, and conducts it to the wet egg-coal screens, while the hopper b_{18}, under the steamboat screen a_{12}, catches the coal dropping through the meshes of this screen, and conducts it into the dry egg-coal screens.

2909. In this view the arrangement of the driving pulleys for the steamboat screens a_{11} and a_{12} can be seen to a better advantage. The pulleys a_1 and a_2 are keyed to the main line shaft V. The circles a_3 and a_4 represent the rope pulleys which are keyed to the same shaft as the pinions a_7 and a_8. These pinions mesh with the screen gears a_9 and a_{10}

of the steamboat screens a_{11} and a_{12}. The deflecting pulleys a_5 and a_6 are used to change the direction of the driving ropes which run over the rope pulleys a_1, a_2, a_3, and a_4.

This view also shows how the gearing for the main elevator u_5 is arranged. The small pinion which gears with the large spur-wheel is keyed to the same shaft as the large rope pulley h_1, which is in line with the driving-rope pulley h keyed to the shaft of the breaker engine. These elevators discharge their contents into a chute that conducts them to the steamboat-screen hopper b_{10}, through which they reach the dry egg-coal screens b_{13} and b_{14}.

2910. The side view of the dry egg-coal screen b_{14} is shown in this cross-section. On the plan another dry egg-coal screen b_{13} is shown, which is parallel to b_{14}, and of the same dimensions. These screens are two of the main screens in the breaker, and are partly double-jacketed. They are driven, as shown, by the small pinion b, under the screen, which meshes with the gear of the screen b_{13}, which in turn meshes with the screen gear b_{10} of screen b_{14}. The pinion b is keyed to the same shaft as the belt pulley b_4, which is in line with the belt pulley b_3 keyed to the main line shaft V.

These screens are supported at the back end by hangers, and at the front end by boxes, which rest on the cap shown in the figure.

The coal that reaches this dry egg-coal screen is comparatively dry, as it is the coal which is cracked up by the different rolls in use in the breaker, and that coming from the lip screens under the breaker.

It must be remembered that the coal that drops through the main screening bars does not reach this dry side—only that part coming from the lip screens under the breaker. That which drops through the meshes of the outside jackets of these screens is everything below the size of chestnut, and is conducted by the hopper c_4 to the dry pea-coal screens f_7 and f_8. The coal coming out of the ends of the outside jackets is chestnut. The coal dropping through the meshes

of the single-jacketed portions of these screens is stove coal, while that coming out of the ends of the screens is egg coal.

2911. The chestnut coal and stove coal coming from these egg-coal screens pass into the cast-iron slate-picker screens d_7 and d_8, shown in this figure by circular rings, which are projections of the circular screen rings to which the slate-picker segments are bolted.

The stove coal coming out of the end of the slate-picker screen d_8 passes into the picking chute *H*. This chute *H* consists of the two supply chutes c_5 and c_6, which receive the coal, slate, and bone, the intermediate chutes c_7 and c_8, where the picking is done, and the delivery chute c_9, which carries off the coal that has been picked over.

The coal coming from the screen, as shown by the arrows, slides down the supply chutes c_5 and c_6, on one side of each of which the intermediate chutes c_7 and c_8 are placed as close to each other as possible, there being room enough c_{10} between each two picking chutes for a man or boy. The delivery chute c_9 is placed so as to receive the coal coming from the intermediate chutes c_7 and c_8. The supply and delivery chutes have the same inclination, but the former is a little the higher, so as to give a slight inclination to the intermediate chutes, the axis of which is placed at an angle of about 8 to 10 degrees with the horizontal, and 25 to 28 degrees with the supply chute.

2912. The intermediate chutes in many cases are specially designed cast-iron chutes, or can, as shown here, be built of plank and sheet iron. The slate picker, who sits with his face towards the upper end of the chute, allows a thin stream of coal to pass in front of him, cleaning it thoroughly as it passes.

The same coal is handled by one man only, with this exception, that one man is placed at the end c_{11} of the delivery chute to inspect the coal and take out any pieces of slate or bone which may have escaped the regular pickers. On each side of the chute *H* are the slate chutes c_{12} and c_{13}, into which the slate pickers throw the slate. These continue to the

bottom, where the slate is examined, to see whether it contains any coal or bone, before it passes into the chutes c_{14} and c_{15}, which lead to the waste pocket *W*, Fig. 1047.

Immediately over the slate chutes, and supported on iron rods, are half round chutes (not shown in the figure), into which the slate pickers throw the bony coal and the pieces that are made up of part slate and part coal. These also continue to the bottom, where they are examined, the pure coal and slate being separated therefrom before they are taken to the bony-coal rolls to be broken.

2913. The chestnut coal coming from the slate-picker screen d_7 is conducted by the rectangular cross-sectioned chute d_{12} to the telegraph w_7, by which it is conducted to the pocket *L*, without being picked by hand, as the greater quantity of the slate has been removed by the slate-picker screen d_7.

The egg coal coming out of the end of the egg-coal screens passes down the chute d_{14} until it reaches the egg-coal picking chute d_{15}, a part of which is cut off in order to show the pea-coal screens f_6 and f_7.

The egg-coal picking chute and the slate and bony-coal chutes are arranged similarly to the stove-coal chute and the slate and bony coal chutes just described.

The material which slides down the hopper c_4, under the egg-coal screens, passes into the dry pea-coal screens f_6 and f_7. The pea screen f_7 is arranged, as shown in the plan, parallel to the screen f_6, which is shown in this view. Both screens are of the same dimensions, and are driven by the small pinion f_4, which meshes with the screen gear f_5 of screen f_7, which in turn meshes with the screen gear f_6 of the screen f_6. The pinion f_4 is keyed to the same shaft as the belt pulley f_3, which is in line with the belt pulley f_2 keyed to the main line shaft *V*. These are double-jacketed screens. The coal coming out of the end of the inside jacket is pea coal, that coming out of the end of the outside jacket is buckwheat, and that dropping through the meshes of the outside jacket into the hopper e_6 is everything below buck-

wheat. The screens are supported at the back end by hangers, and the journals at the front end rest in boxes, supported by the cap e_9.

The pea coal coming out the end of the inside jackets passes down the chute e_{10} to the telegraph, which conducts it to the pea-coal pocket *K*.

The buckwheat coal coming out the end of the outside jackets passes down the chute e_{11} to the telegraph, which conducts it to the buckwheat-coal pocket *J*.

The coal that passes through the meshes of the outside jackets into the hopper e_8 passes into the dry rice-coal screen e_7, which is shown in this cross-section by a number of circular rings, which are the projections of the circular screen rings to which the segments are bolted. This screen is driven, as shown, by the small pinion e_5, which meshes with the screen gear e_6. The rope pulley e_3 is keyed to the same shaft that the small pinion e_5 is keyed to, and is driven by the rope pulley e_2, which is keyed to the main line shaft *V*. The deflecting pulleys e_4 are used to change the direction of the rope that runs over the driving pulleys e_3 and e_2. This is a single-jacketed screen; the coal coming out of the end is rice coal, the smallest size prepared. It passes down the chute e_{12} to the telegraph w_7, which conducts it to the rice-coal pocket *I*. That which goes through the meshes is culm; it passes out into the waste pocket *W*, Fig. 1047.

Part of the coal that drops through the meshes of the three mud screens l, l_1, and l_2, also that which drops through the meshes of the steamboat screen a_{11}, passes into the wet egg-coal screens b_{11} and b_{12}, which are the other two main screens of this breaker, a side view of them being shown in the cross-section.

The screen b_{11} is arranged parallel to b_{12}, and is of the same dimensions. It is driven by the small pinion b_5, which meshes with the screen gear b_7 of screen b_{11}; b_{11} meshes in turn with the screen gear b_6 of the screen b_{12}. The small pinion b_5 is keyed to the same shaft that the belt pulley b_3 is keyed to, and which is in line with belt pulley *1* keyed to the main line shaft *V*.

The wet egg screens b_{11} and b_{12} are both partly double-jacketed screens. The coal that comes out of the end of the outside jacket is chestnut; that which drops through the meshes of the outside jacket into the hopper f_9, under the egg-coal screens, is everything below the size of chestnut; that which drops through the meshes of the single-jacketed portions of the screens is stove coal, and that coming out of the end of the screens is egg coal.

The back ends of these screens are supported by hangers, and the journals at the front end rest in boxes, which are supported by one of the cross-beams. Directly above the screens water troughs are arranged, similar in construction and arrangement to those above the mud screens l, l_1, and l_2.

The coal coming out of the ends of the outside jackets, and that dropping through the single-jacketed portions of the screens, is conducted by the hoppers f_{11} and f_{12} to the two slate-picker screens q_3 and q_4. The back ends of these screens are shown by the circular rings, which are the projections of the circular screen rings to which the slate pickers are fastened.

As shown, these screens are driven by bevel-gearing at the front end of the screens; the small bevel-gear wheels q_2 and q_3 are keyed to the same shaft as the belt pulley q_1, which is in line with the belt pulley q, keyed to the main line shaft V, and shown back of the egg-coal screen by the dotted lines.

The chestnut coal coming out of the end of the slate-picker screen q_4 is conducted to the chestnut-coal jigs p, p_1, and p_2, and the stove coal coming out of the end of the slate-picker screen q_3 is conducted to the stove-coal jigs t and t_1. This view does not show the chute that conducts the flat slate and coal dropping out through the openings in these slate-picker screens and that coming from the slate screens d_7 and d_8 to the No. 5 slate-picker rolls G.

The coal that drops into the hopper f_9 from the egg-coal screens and that coming down the chute b_{16} from the back end of the counter mud screens are run into the wet pea-coal screens y_5 and y_6.

The pea-coal screen y_5 is of the same dimensions as the pea-coal screen y_6, the side elevation of which is shown in this view, and arranged parallel to it. The screen y_5 is driven by the small pinion y_2 on the top of the screen; this pinion meshes with the gear y_3 of the screen y_5, which in turn meshes with the screen gear y_4 of the screen y_6.

The small pinion y_2 is keyed to the same shaft as the belt pulley y_1, which is in line with the belt pulley y keyed to the main line shaft V.

The two wet pea-coal screens y_5 and y_6 are double-jacketed screens. The coal coming out the end of the inside jackets is pea coal, that coming out the end of the outside jackets is buckwheat, and that dropping through the meshes of the outside jackets into the hopper f_{10} is everything below buckwheat.

The pea coal coming out the end of the screens is carried by the chute f_{13} to the pea-coal jig w_1, and the buckwheat coming out of the end of the screens is carried by the chute f_{14} to the buckwheat jig w.

Directly above these screens are troughs, arranged so that they can receive a sufficient quantity of water to thoroughly clean the coal. What passes down the hopper f_{10} under the pea-coal screens y_5 and y_6 passes into the rice-coal screen x_6.

The chutes g_6 and g_7, in front of the rice-coal screen, catch the coal as it comes out of the end of the screen and conduct it to the chute g_8, which carries it to the telegraph w_7, which conducts it to the rice-coal pocket I.

2914. In this view, p, p_1, and p_2 show the front view of the chestnut-coal jigs. They may be divided into two parts, the upper being the machinery for moving the pistons r_2–r_4, and the lower, p–p_2, the jigs proper.

In constructing them, a water-tight tank is built that is divided by partitions which need be only approximately water-tight, as there is water on both sides of them. A detail of the general construction of these jigs is shown in Fig. 1032, with the one exception of the driving gear for the pistons. Here the pistons r_2–r_4 are operated by cams

keyed to the line shaft of the jigs, driven by the pulley r_1 keyed to the same shaft, which is in line with and driven by a pulley keyed to the main line shaft V.

From this view the arrangement of the drag-lines t_3 can be seen; they run over the sprocket-wheels which are driven by the gears o_2 and the rope pulley o_1, which is in line with and driven by the rope pulley o keyed to the main line shaft V.

The jigs t and t_1 are used for stove coal; s_2 and s_3 show the pistons for these jigs, which are driven by the pulley s_1, in line with the belt pulley keyed to the main line shaft V. The drag-lines t_4 for these jigs are driven by the gears u_2 and the rope pulley u_1, which is in line with the pulley u keyed to the main line shaft V. The deflecting rope pulleys u_3 show the position of the pulley that is used to guide the rope that runs over the pulleys u and u_1, above the pea screens y_5 and y_6.

The jigs w and w_1 are for buckwheat and pea coal, and v_2 and v_3 show the pistons for these jigs, which are driven by the pulley v_1, in line with the driving pulley v on the main line shaft V. The drag-lines t_2 and t_3 are for the buckwheat and pea coal jigs, respectively. These drags are driven by the gearing z_2, the small pinion of which is keyed to the same shaft as the rope pulley z_1, which is in line with and driven by the rope pulley z keyed to the main line shaft V of the breaker. The deflecting rope pulleys z_3, which are shown here in relation to the driving pulleys z and z_1, are used to keep the driving rope away from the side of the rice-coal screen x_6.

The stove, pea, and buckwheat coal jigs are similar in construction and arrangement to the chestnut-coal jigs, the details of which are shown in Fig. 1032.

2915. For this wet side the picking chutes for the coal as it comes from the screens and different drag-lines are not shown, but are the same in construction and arrangement as those used on the dry side.

In this cross-section, w_4 shows the level of the floor in the main picking room.

The coal that comes out of the end of the three counter mud screens l, l_1, and l_2 is led by chutes, which are represented in this view by the lines —— - —— and marked h_2, h_3, and h_4. These chutes convey the coal into the main picking chute r (marked r_{11} in Fig. 1047), where it is picked by hand and then passed into the chute h_5, which conducts it to the No. 2, or monkey rolls D, a side view of which is shown in this cross-section.

2916. This view shows the position of the rolls D, the driving pulley n_1, and the gearing. The pulley n_1 is in line with the driving pulley n keyed to the main line shaft V.

The coal, upon leaving the monkey rolls D, passes into the broken-coal screen k, which is shown in this cross-section by a number of circular rings, which are the projections of the circular screen rings to which the segments are bolted. This view shows the miter gear j and the rope pulley i_1, which is in line with the rope pulley i keyed to the shaft of the breaker engine, all of which are used in driving the broken-coal screen k. From this view it can be better understood how the broken coal, as it comes from the end of the broken-coal screen, slides down the chutes leading to the broken-coal drag-line g_6, or to the No. 3 rolls E.

This cross-section shows the side view of the No. 3 or broken-coal rolls E, together with the belt pulley e_1, which is in line with the pulley e keyed to the shaft of the breaker engine.

The broken-coal drag-line g_6, which conveys the coal coming from the broken-coal screen k, is driven by the gearing g_4, which is in turn driven by the pulley g_3; g_3 is in line with the belt pulley g_2 keyed to the shaft that drives the broken-coal screen.

The coal, upon leaving the drag chute h_6, through which it is conveyed, passes into a chute h_7, represented by the line —— - - —— and marked h_7, from which it is picked before it passes into the broken-coal loading pocket O.

The rolls F show the side view of the No. 4 rolls, known as the bony-coal rolls. They are driven by the rope pulley

f_1, which is in line with the rope pulley f keyed to the shaft of the breaker engine.

The rolls G show the side view of the No. 5 rolls, known as the slate-picker rolls. They are driven by the rope pulley g_1, which is in line with the rope pulley g keyed to the shaft of the breaker engine.

As shown, both the No. 4 and No. 5 rolls are on the same level, and the coal coming from them is led by the one hopper to the main elevator u_8. The hopper is shown here by the line —— - —— marked u_8.

2917. In this cross-section is shown the end view of the lump-coal screenings elevator d_{12}. The gearing d_{11} consists of the small pinion gearing into the spur-wheel keyed to the same shaft as the sprocket-wheels over which the elevator-buckets run. This elevator is driven by the pulley d_{10}, which is in line with the pulley d keyed to the shaft d_4, that drives the dry slate-picker screens. As has already been stated, the coal is conveyed to these elevators by a small dump-car running between the lump-coal chute and the elevator.

A side view of the main lip-screenings elevators k_{10} is shown in this cross-section. This elevator, as well as the lump-coal screenings and main elevators, is of the link and bar type.

At the bottom of the breaker a portion of the continuous wall B is removed, so as to show the location of the bottom elevator wheels. These wheels are located below the loading tracks, so that condemned coal can be unloaded from the railroad-car, run into the elevator k_{10}, and hoisted into the breaker to be re-prepared.

The small pinion k_8, the spur-wheel k_9, and the driving pulley k_2, all of which are used in the driving-gear of this elevator, are shown in this view. They are driven by the rope pulley k_1, which is keyed to the driving-shaft of the wet egg-coal screens, the deflecting pulleys k_3 being used to change the direction of the driving rope running over the pulleys k_1 and k_2. The lip-screenings separator k_6 is shown

in this cross-section by two sets of circular rings, which are the projections of the circular screen rings to which the segments are bolted. The small pinion k_4 meshes with the screen gear k_5, and is hidden by the pinion k_6, which is of the same dimensions and keyed to the same shaft. The chutes y_7 and y_8 conduct the buckwheat and pea coal away from the lip-screenings separator k_6.

2918. In this cross-section, a shows an end view of the breaker engine, while b shows the front view of the fly-wheel keyed to the breaker-engine shaft, and c and c_1 show the main belt pulleys keyed to the breaker-engine shaft. These pulleys are in line with the belt pulleys c_2 and c_3 keyed to the main line shaft V.

The steamboat and broken coal coming from the steamboat screens a_{11} and a_{12} is run into the picking chutes r_9 and r_{10}, and picked. These chutes are represented in this view by the line ——- - -—— and marked r_9 and r_{10}.

The steamboat-coal coming from screen a_{11} after it is picked passes into the chute h_{11}, which conducts it to the steamboat loading pocket P.

The chute h_{12} conducts the picked steamboat-coal coming from screen a_{12} into the chute h_{11}. The broken coal coming from the steamboat screen a_{12}, after it has been picked in r_{10}, passes into the chute h_{13}, which conducts it to the chute h_7, where it is re-picked before entering the broken-coal loading pocket O. The broken coal that is picked in chute r_{10}, coming from screen a_{11}, passes into the chute h_{14}, which conducts it to chute h_{13}.

In case there is a limited sale or no sale at all for steamboat and broken coal, the chute h_{15} is so arranged that by sliding a plate in the bottom of the chute h_{11}, the steamboat-coal coming from the two steamboat screens a_{11} and a_{12} will pass down chute h_{16}, and from there into the chute h_6, which leads to the monkey rolls D.

For broken coal there is a similar arrangement in the bottom of the chute h_{13}, which allows the coal to pass into chute h_6, through which it reaches the monkey rolls D.

2919. The pockets for the prepared sizes are sixteen in number, all of which are shown in this view. *I* shows the loading pockets for rice coal, *J* for buckwheat coal, *K* for pea coal, *L* for chestnut coal, *M* for stove coal, *N* for egg coal, *O* for broken coal, and *P* for steamboat-coal.

In the pocket *L* for chestnut coal, on the dry side, is shown the arrangement of the telegraph w_7, which is located in the center of the pocket. All the other telegraphs in the pockets are arranged in the same way. The floors of the pockets are lined with birch boards i_2. The divisions i_3 between the pockets are double-boarded.

In this view one of the loading gates i_4 is shown open, allowing the chestnut coal to pass over the lip screen i_5. The gate i_4 works in cast-iron slides.

The fine coal that drops through the lip screen i_5 is taken up by the drag-line k_{22}, which runs over the sprocket-wheels k_{18} and k_{19}. The drag-line k_{23} running over the sprocket-wheels k_{20} and k_{21} conveys the lip screenings from the wet side. Both of these drag-lines are driven by the pulley k_{11} keyed to the lip-screenings elevator driving-shaft, which is in line with the rope pulley k_{12} keyed to the same shaft as the gear-wheel k_{13}, which gears with the wheels k_{14} and k_{15}, and they in turn drive the bevel-gears k_{16} and k_{17}, which operate the drag-lines k_{22} and k_{23}.

Directly above the lip screens, on both the wet and the dry sides, a trough or pipe is arranged, so that the coal can be washed as it is being loaded into the car for shipment.

In this view all the roofs that would interfere with the view of the machinery have been removed. *Q* shows the roof over a portion of the rock and lump coal chute. *R* and *S* are roofs that cover a portion of the breaker proper, and can be better understood by examining the side view of the breaker in connection with this cross-section.

2920. Lighting the Breaker.—No attempt has been made to show how the breaker is lighted. This is one of the most important items in the construction of a breaker. The main point in the interior arrangement is to have as

much available space as possible, with an abundance of light, especially in the slate-picking rooms, in order to economically and carefully prepare coal for the market.

It has been found by experience that cramped-up breakers are a source of great annoyance and expense in the proper preparation of coal.

There are different ways of arranging window-sashes. Those sliding horizontally will be found very convenient, as they can be most readily opened during the summer months for the purpose of increasing the ventilation. In the picking rooms the windows are generally placed from 5 to 6 feet above the floor level, so that the attention of the slate pickers will not be drawn from their work by any outside attraction.

The platform is another part of the breaker that should be well lighted and ventilated. The parts of the breaker that present any danger to the parties in charge of the machinery should be well lighted. To still further guard against accidents, hand-railings should be placed along the sides of the different walks in the breaker.

2921. Heating.—In the plans of this breaker no method has been shown as to how the breaker is heated. Usually, during the winter months breakers are heated either by steam or by a number of large cast-iron stoves. The latter method of heating is in many cases very unsafe, and to guard as much as possible against fire, it is being replaced by steam heat. Where steam is used, coils of gas-pipe are connected with one or two old cylindrical boilers, the steam being obtained from the different exhaust-pipes of the engines about the colliery. In case it is necessary, a connection, in addition to the above, is made direct with the steam-boilers.

2922. Ventilation.—In breakers where the coal is prepared dry there are large quantities of dust, and in many places the dust is so thick that it is almost impossible for the slate picker to distinguish coal from slate. At such places, small exhaust-fans are sometimes placed, either at the top or the bottom of the breaker, with air-pipes leading to the different parts where the dust is thickest. These fans

draw off the dust from the breaker and discharge it into boxes, where it is dampened by a jet of steam, or into a quantity of water; more frequently, however, they discharge it directly into the open air.

Another method of getting rid of the dust is by stacks leading away from the different rolls to the open air.

In many breakers, and especially in those where water is used, the only method of ventilating is by means of the windows.

THE PREPARATION OF COAL IN THE BREAKER.

2923. Figs. 1046, 1047, and 1048 illustrate this description. In the plan, Fig. 1046, the line

——·——·—— indicates the course of coal during preparation;

——-·——— indicates the course of the prepared coal;

—— ——— indicates the course of the slate and refuse;

——----- indicates the course of the bony coal to be broken.

2924. The mixture of coal, slate, and dirt arrives at the dumps n_6 and a_{11} in mine-cars, where it is dumped; the coal first passes over the main dump-chute bars n_7–n_{10}, which allow most of the smaller coal to pass through into the mud-screen pocket o_3. The coal that does not drop through these bars passes on to the next set of bars n_{11}, so set as to allow a space of 5 inches between each two of them, and that which drops through is led to the main crushers C. Practically all the coal smaller than lump coal passes through these bars, and only lump coal arrives at the platform q_6. The lump coal that arrives on the platform q_6 is divided into three sorts, the first being the slate, which is put into the slate chute q_9, a division in the lump-coal chute; the second consists of those lumps which have been referred to as chippers, containing slate and coal. These lumps are chipped on the platforms q_7; that is, the slate, which can be easily detached from them with a pick, is sent to the slate chute q_9, while

the remaining coal is thrown into the chute a_{14} or a_{16}, Fig. 1048, leading to the main crushers C. The third product consists of more or less pure coal; this is examined by the platform men, and that known as the **rough,** which is not suitable for lump coal, is shoved into one of the holes in the platform opening into the main rolls C. Such lumps as are suitable for lump coal are known as the **glassy,** and are pushed over the plates q_8 to the lump-coal chute q_9, and are loaded out at q_{18} into the railroad-cars for shipment.

2925. All the coal that has gone through the main dump-chute bars q_7–q_{10} is accumulated in the mud-screen hopper or pocket o_2. From this the coal is conveyed to the three mud screens l, l_1, and l_2, the feeding being regulated by means of gates at the bottom of the pocket o_2. The coal that enters these three mud screens is washed by means of the water that pours over the sides of the three troughs erected above the screens, which have already been explained. This washing assists greatly in the separation, more especially in that of the smaller sizes.

In the mud screens l, l_1, and l_2, the first segments extract all sizes below chestnut coal, which are conducted by the chute s_1, Fig. 1047, until they reach the chute b_{10}, Fig. 1048, leading to the hopper f_9, under the egg-coal screens; this hopper then conducts them to the wet pea-coal screens y_5 and y_6.

The other two segments on the mud screens l, l_1, and l_2 take out everything below the size of broken coal, which is conducted by the chute s_6, Fig. 1047, to the chute b_{15}, Fig. 1048, through which it reaches the wet egg-coal screens b_{11} and b_{12}.

2926. The coal coming out of the ends of these three mud screens generally consists of long, flat pieces; this coal must first be broken before it can be put into marketable shape; it is conducted by the chute r_{11}, Fig. 1047, and r, Fig. 1048, to the slate-picking room r_{12}, where men and boys alongside of the chute remove the slate. From this room the

coal passes into the chute h_8, Fig. 1048, which conducts it to the prepared-coal or monkey rolls D.

The coal from the platform q_6, together with that from the bars n_{11}, which has been broken, passes from the rolls C into the roller hopper r_5, which conducts it to the chutes r_6 leading to the two steamboat screens a_{11} and a_{12}. These screens make steamboat and broken coal, and are double-jacketed, the steamboat coming out the end of the inside jacket, and the broken coal out the end of the outside jacket; what drops through the meshes of these outside jackets is everything below the size of broken coal.

The steamboat and broken coal coming out the end of the screens a_{11} and a_{12} is conducted by the chutes r_8 and r_{10} to the slate-picking room r_{12}, where the slate and bony coal are removed. The slate goes to the waste pocket and the bony coal into the chute r_{11}, Fig. 1047 (or r, Fig. 1048), which conveys it to the monkey rolls D. The steamboat-coal that is picked goes to the steamboat pocket P, and the broken coal goes to the broken-coal pocket O, unless there is no sale for the same, in which case it is conducted by the chutes h_{10} and h_{12} to the monkey rolls D, the operation of which has already been described.

The coal that drops through the meshes of the outside jackets on the steamboat screens a_{11} and a_{12} is everything below the size of broken coal, a portion of it going to the dry egg-coal screens b_{13} and b_{14}, and the remainder going to the wet egg-coal screens b_{11} and b_{12}; hence, the coal to be prepared wet in this breaker is all that coming from the three mud screens l, l_1, and l_2 and a portion of the coal smaller than broken that is crushed by the main rolls C.

2927. All this coal, with the exception of that below the size of chestnut, from the mud screens l, l_1, and l_2 passes into the wet egg-coal screens b_{11} and b_{12}; these screens make egg, stove, and chestnut coal. The egg coming out over the end of the screens runs directly to the picking chutes, where the slate and bone are removed by the slate pickers in the main picking room, the floor of which is shown by w in

Fig. 1047 and w_4 in Fig. 1048. From the picking chutes it goes direct to the pocket *N*, from which it is loaded out over the lip screens into the railroad-cars for shipment.

2928. The stove coal coming from these egg-coal screens is conveyed by the hopper f_{11} to the slate-picker screen q_6, where a large part of the flat slate and coal is removed. The coal coming from the slate-picker screen q_6 is conveyed by chutes to the stove-coal jigs t and t_1, where it is washed and the larger part of the slate removed; from these jigs the coal is conveyed by the drag-lines t_4 to the picking chutes, where the very light flat slate that has not been removed, either by the slate-picker screen or jigs, is picked out by the slate-picker boys; they also remove the bony coal that would interfere with the sale of the coal. The coal, after it has been picked, passes directly to the stove-coal pocket *M*.

2929. The chestnut coal comes out the end of the outside jacket of these screens, and is conducted by the chute f_{12} to the slate-picker screen q_4, where a large part of the flat slate and coal is removed. The coal coming from this slate-picker screen is conveyed to the jigs p, p_1, and p_2, which are known as the **chestnut-coal jigs.** Here the coal is washed and the greater part of the slate removed. The coal from these jigs is conveyed by the drag-lines t_6 to a chute which leads directly to the chestnut-coal pocket *L*. That which drops through the meshes of the outside jacket on these egg-coal screens b_{11} and b_{12} is everything below the size of chestnut, and is conveyed with the same product coming from the mud screens l, l_1, and l_2 through the hopper f_9 to the wet pea-coal screens y_5 and y_6, which make pea and buckwheat coal. The pea coal coming out the end of the inside jackets of these screens is conveyed by the chute f_{13} to the pea-coal jig w_1, where it is washed and the greater part of the slate removed.

The coal from this jig is conveyed by the drag-line t_3 to a chute which leads directly to the pea-coal pocket *K*.

2930. The buckwheat coal coming out of the end of the outside jacket is conveyed by the chute f_{14} to the buckwheat-coal jig w, where it is washed and the greater part of the slate removed. The coal from this jig is conveyed by the drag-line t_2 to a chute which leads directly to the buckwheat-coal pocket J.

That which drops through the meshes of the outside jacket on these screens is everything smaller than buckwheat, and is conveyed by the hopper f_{10} to the rice-coal screen x_4 which makes the rice coal. The rice coal that comes out of the end of this screen is conveyed by the chute g_8 to the rice-coal pocket I. What drops through the meshes in this screen is culm or waste, and is conveyed by the water that is used on the screen, through troughs U, Fig. 1046, to the slush bank, where it is deposited.

2931. The large coal coming from the front end of the mud screens l, l_1, and l_2, together with that, when there is no sale for steamboat and broken, coming from the ends of the steamboat screens a_{11} and a_{12}, is all broken up by the monkey rolls D, and is conveyed by the roller hopper w_2 to the broken-coal screen k which makes broken coal. The broken coal coming out the end of the screen slides down a chute leading to the broken-coal drag-line g_5, which conveys the coal to the chute k_7, Fig. 1048, where it is picked before entering the broken-coal pocket O. That which drops through the meshes in the broken-coal screen k is everything below the size of broken coal, and is conveyed by the hopper u_6 to the main elevator u_5. When there is a limited sale, or no sale for broken coal, it is run into the No. 3 rolls E, which break it into sizes below broken coal. These rolls are also so arranged that the coal can be broken into sizes below egg coal. The coal coming from these rolls is conveyed by the roller hopper u to the main elevator u_5.

2932. All the coal that is elevated by the main elevator u_5, and that portion coming from the steamboat screen a_{12}, is conveyed by the hopper b_{16}, under the steamboat screen a_{12}, to the dry egg-coal screens b_{13} and b_{14}. The coal that

reaches these screens is, practically speaking, dry, and in its preparation no water is used until the coal is being loaded into the railroad-cars for shipment; hence, this side of the breaker that is about to be described is the *dry side* (see Fig. 1048).

The dry egg-coal screens, b_{13} and b_{14} prepare egg, stove, and chestnut coal. The egg coal coming out of the end of these screens is conducted by the chutes d_{14} to the picking chute d_{18}, where the slate and bone are taken out before entering the egg-coal pocket *N*.

2933. The stove coal coming from the three segments that compose the single-jacketed portions of these screens is conducted to the slate-picker screen d_8, where the greater part of the flat slate and coal is taken out. The stove coal as it comes from this slate-picker screen passes into the slate-picking chute *H*, where the slate and bony coal are picked out before allowing the coal to enter the stove-coal pocket *M*. The coal coming out the end of the outside jacket of these screens is chestnut coal, and it passes to the slate-picker screen d_7, where the greater part of the flat slate and coal is removed. The coal coming out of the end of this screen is conducted by the chute d_{15} directly to the chestnut-coal pocket *L*. What drops through the meshes of the outside jacket of these screens b_{13} and b_{14} is everything below chestnut coal, and is conducted by the hopper c_4 to the two pea-coal screens f_7 and f_8 which make pea and buckwheat coal. The pea coal coming out of the end of the inside jacket of the screens is conveyed by the chute e_{10} to the pea-coal pocket *K*. The buckwheat coming out of the end of the outside jacket of these screens is conducted by the chute e_{11} to the buckwheat-coal pocket *J*.

What drops through the meshes of the outside jacket of these screens f_7 and f_8 is everything below the size of buckwheat, and is conveyed by the hopper e_8 to the rice-coal screen c_7, which makes rice coal. The rice coal is conveyed by the chute e_{12} to the rice-coal pocket *I*. What drops through the meshes of this screen is culm, or waste, which

goes directly to the waste pocket *W*, Fig. 1047, located underneath the screen.

2934. By examining the plan, Fig. 1046, it will be seen that all the bony coal coming from the picking chutes is conveyed to a point *T* in the breaker by a small dump-car, and dumped into a chute leading to the No. 4 rolls *F*, which are known as the bony-coal rolls. These rolls break the bony coal into sizes below stove coal; the coal is then conveyed by the roller hopper u_6, Figs. 1047 and 1048, to the main elevator u_5, which elevates it, the preparation being performed by passing through the screens on the dry side. In this same plan it is shown that the whole of the flat slate and coal coming from the four slate-picker screens d_1, d_2, q_4, and q_5 is conveyed by chutes to the No. 5 rolls *G*, known as the slate-picker rolls, and is broken up by these rolls into all sizes below chestnut coal. The roller hopper u_8 conveys everything from these rolls to the main elevator u_5; this product is also prepared by passing through the screens on the dry side.

2935. The pieces that drop through the different lip screens, as the coal is being loaded into the railroad-cars for shipment, vary in size, for it will be noticed upon examining the lip screens over which the steamboat-coal passes, that the spaces between the bars are much greater than those over which the chestnut coal is made to pass. All the coal coming from these lip screens is conveyed by the drag-lines k_{22} and k_{23} to the elevator k_{10}. In the drag chutes x_2, through which the drag-lines k_{22} and k_{23} work, are a number of perforated plates which take out the water that is used in washing the coal as it is being loaded into the railroad-cars.

2936. The coal that is elevated by the elevator k_{10} passes into the lip-coal separator k_6 which makes pea and buckwheat coal. The buckwheat coal comes out the end of the outside jacket on this screen, and is conveyed by the chute y_1 to the buckwheat-coal loading pocket *J*. The pea coal drops through the meshes of the single-jacketed portion

of this screen, and is conveyed by the chute y_8 to the pea-coal loading pocket *K*. What drops through the meshes of the outside jacket, and that coming out the end of the screen k_8, is conveyed by the hopper y_9, Fig. 1047, to the main elevator u_8, and receives preparation on the dry side.

In the plan it is shown that the lip-screenings coal coming from the lump-coal chute, that is elevated by the elevator d_{11}, first passes into the broken-coal screen, and finally reaches the main elevator u_8, where it is prepared on the dry side.

2937. The side elevation, Fig. 1047, shows that the slate from the different jigs, and that coming from the different picking chutes, together with the waste coming from the dry rice-coal screen, is all conveyed to the large waste pocket *W*, and is loaded, together with the rock that comes from the platform and collects in the rock chute, in dump-cars that run over the double tracks formed by the rails l_9, l_{10}, and l_{11}.

2938. The refuse coming from the wet rice-coal screen is carried off by troughs, which are indicated in the plan, Fig. 1046, by the line ——— ——— and marked *U*. The water coming from the overflows of the jigs also carries a certain amount of sediment. This water, with that coming from the slush boxes of the jigs, is conveyed to the trough *U* leading to the slush bank.

2939. In Figs. 1046, 1047, and 1048, no attempt has been made to show where the coal that is used in generating steam for the plant is taken from. This will depend upon what size is to be burned. If it is rice coal, it would no doubt be taken from the rice-coal screen e_7 on the dry side; in case it is buckwheat, the buckwheat coal coming from the pea-coal screens f_7 and f_8 would suggest itself; or, as is very often the case, the slate-picker stuff coming from the slate-picker screens is used.

A SERIES

OF

QUESTIONS AND EXAMPLES

RELATING TO THE SUBJECTS
TREATED OF IN THIS VOLUME.

It will be noticed that the questions and examples contained in the following pages are divided into sections corresponding to the sections of the text of the preceding pages, and that each section has a headline which is the same as the headline of the section to which the questions refer. No attempt should be made to answer any questions or to work any examples until the corresponding part of the text has been carefully studied.

GASES MET WITH IN MINES.

EXAMINATION QUESTIONS.

(355) What is chemistry?

(356) What is the difference between an atom and a molecule?

(357) By what means can we determine the volume of gases resulting from any given chemical reaction?

(358) What is meant by the density of air, and at what height above sea-level is it one-half as great as at the sea-level?

(359) Which is the more dense, marsh-gas or carbonic acid gas, and why?

(360) What is specific gravity, and what are the standards or units of measure used (*a*) for solids and liquids? (*b*) for gases?

(361) If a cubic foot of slate weighs 172 pounds, what is its specific gravity? Ans. 2.8.

(362) What is the difference between an element and a compound?

(363) What effect has the force of repulsion on matter?

(364) Describe the principle of the barometer.

(365) What is firedamp, and what are its properties?

(366) What is meant by diffusion of gases?

(367) Name the principal gases formed when a charge of powder is exploded.

(368) What is the difference between mass and volume?

(369) If the specific gravity of anthracite coal is 1.5, what will a cubic yard of it weigh? Ans. 2,531.25 lb.

(370) Is brine, or salt water, a chemical compound or a mechanical mixture?

(371) What is atomic weight, and how does it differ from the ordinary meaning of weight?

(372) What is a chemical equation?

(373) What is a compound substance?

(374) Define the British Thermal Unit.

(375) What is a vacuum?

(376) Why is it difficult to pull two flat wet pieces of glass apart?

(377) What is meant by tension of gases, and what effect has compression on confined gas, the temperature remaining the same?

(378) If 8 cubic feet of air have a tension of 6 pounds per square inch, what will be the volume when the tension is 80 pounds per square inch, the temperature remaining the same? Ans. .6 cu. ft.

(379) What effect has temperature on the volume, when the pressure remains the same?

(380) Of what is marsh-gas a product, and what are its properties?

(381) How is firedamp detected in the mine?

(382) Explain what is meant by "occluded gas."

(383) What is the weight of 300 cubic feet of carbonic acid gas at a temperature of 70° F., barometer 30 inches?

Ans. 34.48 lb.

(384) What are the two principal classes of explosives?

(385) What causes *blown-out* and *windy* shots?

(386) What conditions influence and determine the character of an explosion of gas in a coal-mine?

(387) What is a safety-lamp? State the principle of its action.

(388) Upon what does the force of a gas explosion depend?

(389) What produces white damp, what is its effect on human life, and how may it be detected?

(390) What are the most common causes of the ignition of gas in mines?

(391) What are the Sprengel explosives, and what peculiar good feature do they possess?

(392) What causes tend to lessen the heat of an explosion?

(393) If 20 cubic feet of air at a temperature of 60° F., and a pressure of one atmosphere, are compressed to 12 cubic feet (the temperature still remaining at 60° F.), what will the compressed air weigh per cubic foot? Ans. .1272 lb.

(394) Name the three general classes of detonating explosives.

(395) If a volume of 3 cubic feet of air is under a pressure of 36 pounds per square inch, what will the pressure be when the volume is increased to 4 cubic feet?

Ans. 27 lb. per sq. in.

(396) What causes sudden outbursts of gas?

(397) Which is the better blasting powder: that composed of potassium nitrate, sulphur, and carbon, or that composed of sodium nitrate, sulphur, and carbon?

(398) What is spontaneous combustion?

(399) How many kinds of barometers are there?

(400) Give the composition of the atmosphere about us.

(401) What is oxidation?

(402) How many B. T. U. are produced by burning 2,000 pounds of bituminous coal? Ans. 28,800,000 B. T. U.

(403) What is the difference between deflagration and detonation?

(404) What is the result of combustion?

(405) Explain the result of mixing two liquids which do not act chemically upon each other, and which have different densities; also, explain the result of mixing two gases which do not act chemically upon each other, and which have different densities.

(406) Give the symbols of (*a*) water; (*b*) marsh-gas; (*c*) carbonic acid gas; (*d*) carbonic oxide gas. State the advantage of adopting such a system of symbols.

(407) What is atomic volume?

(408) What weight of carbonic acid gas will be produced by burning 300 pounds of coal containing 88% of carbon?

Ans. 968 lb.

(409) What is dissociation?

(410) Explain (*a*) the force that binds atoms together; (*b*) the force that binds molecules together.

(411) A piece of iron weighs 10 pounds when weighed in air, and 8.6 pounds when weighed in water; what is its specific gravity? Ans. 7.14.

(412) To what do the first and second laws of volume apply?

(413) If a volume of 1,200 cubic feet of marsh-gas is mixed with pure air in such a proportion that when exploded all the carbon in the marsh-gas combines with the oxygen in the air, what volume of carbonic acid gas will result, the carbonic acid gas and the marsh-gas being subjected to the same pressure and temperature? Ans. 1,200 cu. ft.

(414) What are the three forms of matter?

(415) What are the properties of sulphureted hydrogen gas, and how is the presence of the gas detected?

(416) What causes *feeders* or *blowers* of gas?

(417) In what ways do various kinds of coal-dust influence the character of an explosion?

(418) Describe the Davy lamp.

(419) What is the most practical manner of detecting firedamp in mines?

(420) How is the percentage of gas in the atmosphere measured by a Pieler lamp?

(421) What must be the volume of a vessel to hold two gases whose volumes are 15 cubic feet and 9 cubic feet, and whose tensions are 20 pounds and 15 pounds, respectively,

in order that the pressure of the mixture may be 21 pounds per square inch, the temperature remaining the same throughout? Ans. 20.71 cu. ft.

(422) What effect has a small amount of carbonic acid gas upon the flame-cap of a safety-lamp?

(423) Explain the manner in which a test is made for firedamp with a safety-lamp.

(424) What is black damp, what produces it, what is its effect on human life, and how can it be detected?

(425) Describe the Shaw gas-testing machine.

(426) What oils are generally used in safety-lamps?

(427) State the requirements of a good safety-lamp (*a*) for general mining use; (*b*) for testing purposes

(428) What is nitroglycerine, and what ruptive pressure is exerted by it when exploded?

(429) If a quantity of air confined in a cylinder and at a temperature of 50° F. is heated until it has a temperature of 212° F., what will be the resulting tension if the original tension was 14.7 pounds per square inch?

Ans. 19.38 lb. per square inch.

(430) State the conditions under which mine explosions are most frequently produced.

(431) In what respect does the Clanny lamp differ from the Davy lamp?

(432) Why is it that the Mueseler lamp gives a better light than either the Marsaut lamp or Clanny lamp?

(433) Describe the Ashworth-Hepplewhite-Gray lamp.

(434) What is dynamite, and what proportion of nitroglycerine is contained in each grade?

(435) If a volume of 76 cubic feet of confined air weighs 6.5 pounds and has a temperature of 84° F., what pressure per square inch does it exert?

Ans. 17.21 lb. per square inch.

(436) In what respects is the Evan Thomas lamp better than the Clanny lamp?

(437) What peculiar feature is possessed by the Marsaut lamp, and how may its illuminating power be improved?

(438) What is guncotton, and how is it exploded?

(439) What are (*a*) tonite? (*b*) potentite? (*c*) gelatine-dynamite?

(440) When gas is exploded, what effects are caused by coal-dust suspended in the air?

(441) Upon what does the strength of an explosive depend, and which is the best type for use in working non-gaseous coal-mines?

(442) If the specific gravity of marsh-gas at a temperature of 60° F., barometer 30 inches, is 0.559, what will 100 cubic feet of it weigh? Ans. 4.28 lb.

(443) Why does black damp diffuse in air slower than firedamp?

(444) What is the weight of 650 cubic feet of marsh-gas at a temperature of 60° F., the barometer being at 29.5 inches? Ans. 26.75 lb.

MINE VENTILATION.

(PART 1.)

EXAMINATION QUESTIONS.

(445) Define (*a*) gravitation; (*b*) the mass of a body; (*c*) acceleration.

(446) If a cannon-ball is shot vertically upwards with an initial velocity of 1,876 feet per second, (*a*) what time will elapse before it reaches the ground again? (*b*) What distance will it travel?

Ans. { (*a*) 1.944 min., nearly. (*b*) 109,433.3 ft.

(447) What are the requirements that should be considered in fixing the quantity of air for any particular mine?

(448) State the quantity of air per man, per minute, required by law (*a*) in the anthracite region; (*b*) in the bituminous regions of the Union.

(449) When the velocity of the current is 300 feet per minute, what quantity of air is passing through a 7 ft. × 7 ft. airway? Ans. 14,700 cu. ft. per min.

(450) If a water-gauge of 2 inches passes 15,000 cubic feet of air per minute, what quantity per minute will a water-gauge of 8 inches pass in the same airway?

Ans. 30,000 cu. ft.

(451) If 2 horsepower pass 14,000 cubic feet of air per minute, to what must the power be increased to double the quantity? Ans. 16 H. P.

(452) What is the total ventilating pressure in an airway 6 ft. × 7 ft., the water-gauge being 1.5 inches?

Ans. 327.6 lb.

(453) If you have two airways under the same pressure, one 6 ft. × 6 ft. × 5,000 ft., and the other 8 ft. × 4½ ft. × 5,000 ft., which will pass the greater quantity of air, and why?

(454) In a certain mine the total quantity of air passing down the downcast shaft is 45,000 cubic feet per minute. At the foot of the downcast it is divided into four splits as follows: Split (1) 6 ft. × 6 ft., 1,500 feet long; Split (2) 6 ft. × 7 ft., 1,800 feet long; Split (3) 6 ft. × 5 ft., 1,350 feet long; Split (4) 5 ft. × 5 ft., 1,500 feet long. Calculate the amount of air passing in each split when no regulators are used.

Ans. { 12,582 cu. ft. per min. for (1).
13,905 cu. ft. per min. for (2).
10,539 cu. ft. per min. for (3).
7,974 cu. ft. per min. for (4).

(455) If the current in an underground road 6 feet square is maintained by a pressure represented by 1 inch of water-gauge, what pressure per square foot will be required to pass the same quantity of air along a road 5 feet square, the two roads being of the same length? Ans. 12.94 lb. per sq. ft.

(456) If the velocity of an air-current is 4 feet per second, and it is required to increase it to 8 feet per second, what will be the ratio of increase in the power?

(457) If the airways of a mine were increased to double their length, other conditions remaining the same, in what proportion would you have to increase the ventilating pressure?

(458) In order to obtain double the quantity of air, in what proportion must the ventilating pressure be increased?

(459) In question 458, in what proportion would the power have to be increased to obtain the same result?

(460) The anemometer shows a current of 10,000 cubic feet of air per minute to be passing down the intake of a mine where the temperature is 30° F. Supposing no increase of the flow from the gases of the mine, what quantity of air will be passing per minute up the return where the temperature has risen to 70° F.? Ans. 10,818 cu. ft. per min.

(461) The quantity of air produced by a fan is 120,000 cubic feet per min. and the water-gauge is 2 inches. What is the horsepower absorbed in the mine? Ans. 37.82 H. P.

(462) If a water-gauge of 3 inches passes 20,000 cubic feet of air per minute in a certain mine, what water-gauge will be required to pass 30,000 cubic feet per minute through the same mine under similar conditions? Ans. 6¾ in.

(463) A current of 60,000 cubic feet of air per minute is circulated in a certain mine in five splits as follows: Split No. 1, 8,000 cu. ft.; Split No. 2, 10,000 cu. ft.; Split No. 3, 12,000 cu. ft.; Split No. 4, 14,000 cu. ft.; Split No. 5, 16,000 cu. ft. Calculate the sectional area for each split, in order that the air may travel at a uniform velocity of 5 feet per second in all the splits.

Ans. $\begin{cases} \text{No. 1, } 26\frac{2}{3} \text{ sq. ft.} \\ \text{No. 2, } 33\frac{1}{3} \text{ sq. ft.} \\ \text{No. 3, } 40 \text{ sq. ft.} \\ \text{No. 4, } 46\frac{2}{3} \text{ sq. ft.} \\ \text{No. 5, } 53\frac{1}{3} \text{ sq. ft.} \end{cases}$

(464) The quantity of air passing per minute in a mine is 112,000 cubic feet, the effective power of the furnace is 40 horsepower. Required, the height of the water-gauge in inches. Ans. 2.27 in.

(465) A current of 10,000 cubic feet of air per minute is passing into a mine having two splits as follows: Split *A*, 4 ft. × 12 ft., 6,000 feet long; Split *B*, 6 ft. × 8 ft., 10,000 feet long. Find the amount of air passing through each split.

Ans. $\begin{cases} A\text{, 5,470 cu. ft. per min.} \\ B\text{, 4,530 cu. ft. per min.} \end{cases}$

(466) Suppose 50,000 cubic feet of air per minute to be passing in four splits, as follows:

Split *A*, 6 ft. × 8 ft., 10,000 ft. long, and 5,000 cu. ft. per min.

Split *B*, 5 ft. × 10 ft., 5,000 ft. long, and 10,000 cu. ft. per min.

Split *C*, 6 ft. × 12 ft., 10,000 ft. long, and 20,000 cu. ft. per min.

Split *D*, 4 ft. × 12 ft., 5,000 ft. long, and 15,000 cu. ft. per min.

(*a*) In which splits would regulators have to be introduced to accomplish such division of air, and why? (*b*) What is the entire horsepower required for the circulation in the splits? Ans. (*b*) 12.685 H. P.

(467) What is the height of the motive column if the water-gauge reads .4 inch? The temperature in the downcast shaft is 62° F., and the barometer is 30 inches.
Ans. 27.256 ft.

(468) If the air in a mine has a temperature of 62° F., and is kept in motion by a ventilating pressure of 2.08 pounds per square foot, what will be the theoretical velocity of the air, the barometer being 30 inches?
Ans. 2,512.38 ft. per min.

(469) If the water-gauge reads .6 inch when 15 horsepower is used, what will it read when the power is increased to 36 horsepower? Ans. 1.076 in.

(470) If a ventilating current of 10,000 cubic feet of air per minute passes through an airway 6 ft. × 8 ft. and 8,000 feet long, what current will pass when the length of the airway is increased to 10,000 feet?
Ans. 8,945 cu. ft. per min.

(471) 24,000 cubic feet of air per minute are passed through an airway 8 ft. × 10 ft. and 5,000 feet long. What should be the dimensions of a similar airway 8,000 feet long to pass the same quantity? Ans. 8.8 ft. × 11 ft.

(472) A square airway has a sectional area of 64 square feet, and has passing through it 15,000 cubic feet of air per minute. What must be the sectional area of a similar airway to pass 20,000 cubic feet per minute, the power remaining the same? Ans. 90.38 sq. ft.

(473) An airway is passing 20,000 cubic feet of air per minute, and it is desired to reduce this quantity to 8,000 cubic feet by means of a regulator. If the difference of pressure on the two sides of the regulator is equivalent to $\frac{1}{2}$ inch of water-gauge, what must be the area of the opening? Ans. 6.4 sq. ft.

(474) An airway 8 ft. × 10 ft. passes 60,000 cubic feet of air per minute to a point 1,500 feet distant from the downcast shaft, where it splits into four airways of the following dimensions: 1st, 6 ft. × 5 ft., 900 feet long; 2d, 6 ft. × 6 ft., 825 feet long; 3d, 6 ft. × 4 ft., 840 feet long; and 4th, 5 ft. × 4 ft., 720 feet long. (*a*) What is the quantity passing through each split, and (*b*) what should be the reading of the water-gauge for the mine, neglecting the shafts?

Ans. (*a*) 1st. 15,942 cu. ft. per min.; 2d. 20,952 cu. ft. per min.; 3d. 12,381 cu. ft. per min.; 4th. 10,725 cu. ft. per min. (*b*) 2.36 in.

(475) How much (i. e., in what proportion) must the ventilating power be increased to double the quantity of air?

(476) In a certain mine the entire circulation of 100,000 cubic feet of air per minute is divided into two splits having equal cross-sections. If the resistances of the splits are to each other as 5 is to 1, what quantity of air will pass through each?

Ans. 69,099 cu. ft. per min. in short airway. 30,901 cu. ft. per min. in long airway.

(477) The depth of a furnace shaft is 300 feet; the temperature of the upcast column of air is 130° F., and that of the downcast column, 50° F. (*a*) What is the length of the motive column? (*b*) What would be the pressure producing ventilation should the temperature of the upcast be raised to 150° F., while that of the downcast remained at 50° F.? Assume the barometer reading to be 30 inches.

Ans. (*a*) 40.13 ft. (*b*) 3.85 lb. per sq. ft.

(478) Suppose that 50,000 cubic feet of air per minute pass through an airway 10 ft. × 10 ft. and 10,000 feet long, and that a change is made by dividing the current into three splits of the following dimensions: 1st, 6 ft. × 6 ft. and 4,000 feet long; 2d, 5 ft. × 6 ft. and 3,000 feet long; 3d, 5 ft. × 5 ft. and 4,000 feet long. (*a*) What quantity will pass through each of the splits that are now substituted for

the original airway, assuming that the total quantity remains the same? (*b*) What horsepower is required in each case?

Ans. (*a*) 1st. 19,597 cu. ft. per min.; 2d. 17,980 cu. ft. per min.; 3d. 12,423 cu. ft. per min.
(*b*) 1st case, 32.88 H. P.; 2d case, 25.98 H. P.

(479) If an airway 9 ft. × 6 ft. has a total rubbing surface of 54,000 square feet, what is its length? Ans. 1,800 ft.

(480) What quantity of air is passing through an airway 7.25 ft. × 11.75 ft., when the velocity is 434 feet per minute? Ans. 36,971 cu. ft. per min.

(481) In what proportion must the ventilating pressure be increased to change the volume of air from 120,000 cubic feet to 180,000 cubic feet per minute? Ans. $2\frac{1}{4}$ times.

(482) If a pressure of 19 2 pounds per square foot passes 160,000 cubic feet per minute, what pressure is required to pass 120,000 cubic feet per minute through the same airway? Ans. 10.8 lb. per sq. ft.

(483) What horsepower is required to pass 70,000 cubic feet of air per minute when the water-gauge reading is .9 inch? Ans. 9.927 H. P.

(484) The upcast shaft is 300 feet deep, and the temperature of the ascending air is 120° F. The temperature of the downcast shaft being 45° F., what is the height of the motive column? Ans. 38.86 ft.

(485) If .6 inch of water-gauge passes 12,000 cubic feet of air per minute, what pressure per square foot is required to pass 24,000 cubic feet? Ans. 12.48 lb. per sq. ft.

(486) If a pressure of 4 pounds per square foot produces a velocity of 300 feet per minute in a 6 ft. × 8 ft. airway, what pressure is required to pass 24,000 cubic feet per minute through the same airway? Ans. $11\frac{1}{9}$ lb. per sq. ft.

(487) If 20,000 cubic feet of air are passed per minute

by a pressure of $2\frac{1}{2}$ pounds per square foot, what must be the horsepower to pass 25,000 cubic feet per minute?

Ans. 2.959 H. P.

(488) (*a*) What is the area of a square airway through which a current of 30,000 cubic feet of air per minute is passing at a velocity of 500 feet per minute? (*b*) What should be the area of each of two square airways to divide the current into two equal splits, the length of the airways and the velocity of the air being the same in both cases? (*c*) Which of these arrangements requires the greater power, and in what proportion?

Ans. (*a*) 60 sq. ft.
(*b*) 30 sq. ft.
(*c*) The two small airways, in the ratio of 1.4194 : 1.

(489) If 24,000 cubic feet of air per minute pass through a 6 ft. × 10 ft. airway, what quantity will pass through a 5 ft. × 6 ft. airway of the same length, the power remaining the same? Ans. 13,596 cu. ft. per min.

(490) If 40,000 cubic feet of air per minute are passing in a circular airway 8 feet in diameter and 1,800 feet long, what is (*a*) the pressure per square foot? (*b*) the horsepower required?

Ans. (*a*) 12.36 lb. per sq. ft.
(*b*) 14.982 H. P.

(491) A 7 ft. × 10 ft. airway is passing 35,000 cubic feet of air per minute, and it is desired to reduce this quantity to 21,000 cubic feet per minute by means of a regulator. The water-gauge reading $\frac{3}{4}$ inch, what must be the area of the opening in the regulator? Ans. 12.12 sq. ft.

(492) What do you understand by the terms (*a*) motive column? (*b*) ventilating pressure? (*c*) split?

(493) (*a*) What is the chief use of the regulator? (*b*) To what is it equivalent? (*c*) What are the effects produced by splitting the air, and what advantages are obtained by splitting?

(494) The water-gauge at a mine is .7 inch. If the length of the airway is increased to three times its original

length, and the velocity is increased from 8 to 10 feet per second, what will the water-gauge read? Ans. 3.28 in.

(495) There are two splits in a mine, one 2,000 yards long and the other 5,000 yards long; each has a section 7 ft. × 6 ft. (*a*) With a water-gauge of $2\frac{1}{2}$ inches for underground friction, what quantity will pass through each split? (*b*) What horsepower is required for the two splits? (*c*) What area of regulator opening will be required to reduce the quantity in the short split one-half?

Ans. (*a*) 1st. 16,868 cu. ft. per min.; 2d. 10,668 cu. ft. per min.
(*b*) 10.85 H. P.
(*c*) 2.46 sq. ft.

(496) If 50,000 cubic feet of air pass in a circular airway 18 feet in diameter, what quantity will pass in an airway 6 feet in diameter, the power remaining the same?

Ans. 8,012.5 cu. ft. per min.

(497) (*a*) What are similar figures? (*b*) Define the coefficient of friction as used in mine ventilation. (*c*) Define power and work.

(498) Extract the fifth root (*a*) of 35, and (*b*) of 64,268,937, by means of the rule given in Art. **1000**.

Ans. (*a*) 2.036.
(*b*) 36.44.

MINE VENTILATION.

(PART 2.)

EXAMINATION QUESTIONS.

(499) By what two methods is air set in motion so as to produce a current?

(500) Mention the different means at hand for producing ventilation in mines.

(501) (*a*) What conditions are necessary to produce natural ventilation in a mine? (*b*) How does this form of ventilation differ from all others?

(502) Explain the use of the furnace. Where is it always placed, in the ventilation of a mine?

(503) The depth of a certain upcast shaft is 500 yards; the mean barometric pressure is 30.25 inches, and the mean temperature of the air in the shaft is 350° F. What is the weight of a column of air in this shaft, having a base of 1 square foot? Ans. 74.25 lb.

(504) What is the weight of a cubic foot of air when the barometric reading is 29.3 inches and the temperature is 32° F.? Ans. .0791 lb., nearly.

(505) If the anemometer records a velocity of 800 feet per minute in the intake airway of a mine, where the sectional area measures 8′ × 10′ and the thermometer shows a temperature of 32° F., what should be the volume of air passing in this same airway per minute, at a point where the temperature has risen to 60° F.? Ans. 67,650 cu. ft., nearly.

(506) The depths of the downcast and upcast shafts are each 540 feet. Their respective average temperatures are 0° F. and 300° F. Calculate the pressure per square foot

that produces ventilation in this mine when the average barometric pressure is 29.8 inches. Use two methods.

Ans. 18.365 lb.

(507) What is the height of motive column which produces ventilation, when the depths of the upcast and downcast shafts are each 200 yards, and their respective average temperatures are 60° F. and 360° F.? Find the motive column in the downcast shaft. Ans. 219.78 ft.

(508) Mention some of the essential points in the construction of a mine furnace.

(509) What will be the area of fire-grate of a furnace which is to supply 50,000 cubic feet of air per minute against a 2-inch water-gauge when the depth of the furnace shaft is 250 feet? Ans. 33.88+ sq. ft.

(510) (*a*) Explain the object of a dumb drift. (*b*) What is the minimum height above the furnace of the point where a dumb drift may enter a shaft with safety?

(511) Describe the method of ventilating by means of a waterfall.

(512) (*a*) What do you understand by a mechanical ventilator? (*b*) Mention some examples of such ventilators.

(513) What are the most prominent types of centrifugal ventilators now in use?

(514) (*a*) In what two ways do centrifugal fans act? (*b*) What can you say of the relative efficiencies of these two modes of action?

(515) Describe the general characteristics of the Waddle fan.

(516) What are some of the chief points in the construction of the Schiele fan?

(517) What is the effect of the spiral casing surrounding the circumference of a ventilating fan?

(518) What is the purpose of the évasée chimney?

(519) Describe the general form of the Guibal fan.

(520) What can you say of the Capell fan?

(521) In what two respects is the fan a better means of ventilation than the furnace?

(522) Plainly stated, what is the difference between the action of the furnace and the action of the fan?

(523) What is the velocity of air blowing through an orifice into a vacuum under a pressure of 8.41 pounds per square foot? Ans. 52.2 ft. per sec.

(524) What is the velocity at which air should enter the fan to produce the best results?

(525) A fan is designed to produce a current of 175,000 cubic feet of air per minute. What should be the diameter of its central orifice if it receives its air upon each side?
Ans. 10.146 ft.

(526) (*a*) What do you understand by the throat of a fan? (*b*) What determines the area of the throat of a fan?

(527) What should be the width of blade in a fan designed to throw 250,000 cubic feet of air per minute? The fan receives its air upon each side. Ans. 6.06 ft.

(528) The area of the port of entry of a fan which receives its air upon one side is 153.9384 square feet. What should be the breadth of its blades? Ans. 3.5 ft.

(529) What do you understand by manometric efficiency as relating to fans?

(530) Name three essential elements to the efficient ventilation of a mine.

(531) How is the quantity of air that is necessary for the ventilation of a mine determined, and what amount is customary in non-gaseous mines? What amount is usually specified for a gaseous mine?

(532) In order to secure thorough ventilation in a mine, what is necessary with respect to the *velocity* of the air-current?

(533) (*a*) What danger arises in gaseous mines from too high a velocity? (*b*) What is the maximum velocity allowed under the Anthracite Mine Law of Pennsylvania?

(534) What means are necessary in order to properly conduct an air-current to the working face? Mention, also, the essential points necessary to be observed with respect to each of these means.

(535) What instruments are used for measuring the resistance of airways, and what does each measure, respectively?

(536) Describe the water-gauge and the manner of using it for determining the ventilating pressure in a mine.

(537) Describe the anemometer and the manner of using it, stating also what precautions are necessary in order to obtain an average velocity for the entire area of the airway.

(538) Upon what does the density of air mainly depend, and what instruments are used in determining the density (the weight of a cubic foot) of air?

(539) (*a*) Name the freezing and boiling points of the Centigrade and Fahrenheit scales, respectively. (*b*) How many degrees of the Fahrenheit scale correspond to 100° of the Centigrade?

(540) Convert (*a*) 350° C. into the corresponding Fahrenheit reading; (*b*) – 10° C.; (*c*) – 25° C.

Ans. (*a*) 662° F. (*b*) 14° F. (*c*) – 13° F.

(541) What readings of the Centigrade scale correspond to the following: (*a*) 365° F.; (*b*) 5° F.; (*c*) – 49° F.?

Ans. (*a*) 185° C. (*b*) – 15° C. (*c*) – 45° C.

(542) To what is the pressure *per square foot* of sectional area in an airway, due to any air column, always equal?

(543) Give examples of vertical and inclined air columns in the practical ventilation of mines, and state how the pressure per square foot at the base of the inclined column is calculated.

(544) Which class of workings is more easily ventilated, *rise* or *dip* workings, and why?

(545) What is meant by (*a*) positive air columns? (*b*) negative air columns?

(546) What do you understand by ascensional ventilation?

(547) To what is the algebraic sum of the weights of all the columns, positive and negative, equal?

(548) What is the main feature in the ventilation of a flat, non-gaseous seam?

(549) What is the practical limit to the splitting of air-currents?

(550) How does the presence or absence of gas affect the arrangement of the haulage roads with respect to the ventilating current?

(551) What is the main point to be considered in the ventilation of inclined seams?

(552) Mention some important points in connection with the entrance of a mine after an explosion.

(553) Name the different methods of treating mine fires.

(554) What precaution is necessary to be taken in building stoppings for the isolation of a mine fire? Explain the reason why such precaution is necessary.

HOISTING AND HOISTING APPLIANCES.

EXAMINATION QUESTIONS.

(1585) What is the motor of a hoisting plant?

(1586) Name the motors that fill the requirements of the hoisting service.

(1587) Explain the two methods according to which engines are used as motors for a hoisting plant. Which of these methods is usually adopted?

(1588) State. the advantages of the electric motor for hoisting purposes.

(1589) What are the objections to a direct current for long distance work? Explain how the alternating current overcomes this objection.

(1590) Explain the principle of the multiphase synchronous motor. What are its advantages for hoisting purposes?

(1591) Describe a speed controller for an electric hoist.

(1592) What is a duplex engine?

(1593) Explain why a single-cylinder engine will not fulfil the requirements of the hoisting service, and how a duplex engine will fulfil those requirements.

(1594) (*a*) What is to be gained by the use of condensing engines? (*b*) What are the objections to condensing engines?

(1595) If a condensing engine is to be used for hoisting, what provisions must be made?

(1596) Why should a condensing engine for hoisting purposes be cross-compound?

(1597) What is initial condensation? Why should the initial condensation be greater in a single-cylinder condensing engine than in a single-cylinder non-condensing engine?

(1598) Explain why a compound hoisting-engine should have an auxiliary steam-pipe and throttle-valve to admit live steam to its low-pressure cylinder.

(1599) Explain why a condensing hoisting-engine should have an independent air-pump and condenser.

(1600) What is meant by the size of an engine?

(1601) What kinds of work constitute the work required of a hoisting-engine?

(1602) In a hoisting plant, what items make up the load on the rope?

(1603) How small a drum is it safe to use with a pliable steel rope?

(1604) (*a*) What is the net load on a hoisting-engine? (*b*) What is the gross load? (*c*) What is the actual load? (*d*) Give a rule for determining the actual load.

(1605) If we have a shaft or vertical hoistway 1,200 feet deep and wish to hoist from it 3,000 pounds of material at a trip, in a car weighing 1,800 pounds and a cage weighing 2,400 pounds, (*a*) how large a plow-steel rope, 19 wires to the strand, should we use? (*b*) How small a drum would it be safe to wind this rope on?

Ans. { (*a*) 1 in. (*b*) 5 ft.

(1606) What would be the actual load on the engine, in Question 1605, if the shaft were double, so that two cages may be used to balance each other, and if no tail-rope were used? Ans. 6,225.6 lb.

(1607) Give a rule for finding the work required of a hoisting-engine per revolution of the drum.

(1608) In example 1606, if the smallest drum advisable were used, what would be the work required of the engine per revolution of the drum. Ans. 99,421.4 ft.-lb.

(1609) What should be the size of the engine cylinders to do the hoisting in the case mentioned in Question 1606? Assume the M. E. P. to be 48.76 pounds per square inch and the ratio of the stroke to the diameter $1\frac{1}{2}$.

Ans. 22 in. × 33 in.

(1610) If the cylinder of an engine is 12 inches in diameter and the stroke is 24 inches, what work per revolution will it perform if supplied with steam at 40 pounds M. E. P.?

Ans. 18,096 ft.-lb.

(1611) In allowing for the extra force necessary to overcome friction and accelerate the moving parts, why is a certain percentage of the gross load taken instead of a larger percentage of the net load?

(1612) (*a*) What should be the size of the cylinders of a duplex hoisting-engine which performs 36,000 foot-pounds of work per revolution, if it is supplied with steam at 40 pounds M. E. P., and if the stroke is to be $2\frac{1}{2}$ times the diameter? (*b*) What is the least turning moment that this engine will exert?

Ans. { (*a*) 14 in. × 35 in.
(*b*) 107,757 in.-lb.

(1613) How do you determine the turning moment exerted by the load on the drum?

(1614) If a tail-rope is used, must the engine be larger than when no tail-rope is used, or may it be smaller?

(1615) If a shaft be made single instead of double, thus preventing the use of two cages and two cars balancing each other, will the engine need to be larger or smaller?

(1616) State the relative advantages of steam and compressed air for hoisting-engines, and state the conditions which influence the choice between steam and air.

(1617) Why should hoisting-engines be equipped with cylinder relief-valves? Explain the principle of the relief-valves illustrated in Fig. 892.

(1618) What are the three principal kinds of drums?

(1619) If we have a shaft 1,800 feet deep and wish to hoist from it such a load as would, with the weight of the rope, require a rope $1\frac{1}{2}$ inches in diameter, how long a parallel drum must we have, allowing $\frac{1}{4}$-inch between the coils?

Ans. 11 ft. 8 in.

(1620) What is the object of using conical instead of cylindrical drums?

(1621) (*a*) How would you determine the small diameter of a conical drum? (*b*) Give a rule for determining the large diameter of a conical drum.

(1622) Should metal ropes for hoisting purposes be allowed to coil upon themselves? Why? What is the chief objection to the use of flat ropes?

(1623) If we wish to build a pair of conical drums for a vertical shaft 800 ft. deep, from which it is desired to hoist 4,000 lb. of material at a trip, in cars weighing 3,000 lb. each and with cages weighing 3,200 lb. each, what should the large and small diameters be? The rope is to be of cast steel, 19 wires to the strand.

Ans. $\begin{cases} \text{Large diameter} = 9 \text{ ft.} \\ \text{Small diameter} = 7 \text{ ft.} \end{cases}$

(1624) What are the objections to both cylindrical and conical drums?

(1625) Describe the Whiting system of hoisting. What are its advantages?

(1626) (*a*) What is the use of a brake of a hoisting plant? (*b*) What load may be brought upon the brake?

(1627) What is a block brake? Describe the strap brake.

(1628) Why should the ends of a brake strap be kept close to the drum?

(1629) In the case of a very large strap brake, how would you keep the upper half away from the drum when the brake is off?

(1630) Why can not the force that a man is able to exert be so multiplied by levers as to be sufficient for any brake?

(1631) Show what can be accomplished by a differential leverage, and explain the principle involved.

(1632) Having the force that is to be applied to the end of a brake strap, how can we determine the resistance that the strap offers to the rotation of the drum?

(1633) If the hoisting plant is so large that a man can not do the braking by hand, what other agents may be used? How can these agents be applied to a brake-gear?

(1634) What is a hoist indicator? Describe a positive-motion indicator.

(1635) Explain the Kœpe system of hoisting.

(1636) Explain a method of capping a hoisting rope.

(1637) What is a detaching-hook, and why is it used?

(1638) What is a tail-rope? What is gained by using a tail-rope?

(1639) What is a cross-head for mining purposes? For what is it used?

(1640) (*a*) Why is a cross-head made to rest on the rope without being attached to it? (*b*) Explain a way to make such an arrangement.

(1641) What is a cage, and what is its use?

(1642) What two general kinds are there?

(1643) Explain the action of the usual kind of safety-catches for shaft cages.

(1644) What are gunboats, or skips?

(1645) What are rope carriers used for?

(1646) What are sheaves, and for what are they used?

(1647) What are the advantages of a sheave with wrought-iron spokes over one with cast-iron arms?

(1648) Explain the method of putting in wrought-iron spokes tangentially.

(1649) Describe a method of dumping a gunboat, or skip.

(1650) What are landing fans, or keeps?

(1651) How would you find the direction and amount of the stress to which a head-frame was subjected? Explain fully.

SURFACE ARRANGEMENTS
OF
BITUMINOUS MINES.

EXAMINATION QUESTIONS.

(1652) How does a small seam and bad roof influence the size of the mine-car?

(1653) At a shaft mine with two openings, how should the machinery for hoisting, ventilation, pumping, and underground haulage and coal cutting be grouped around these openings?

(1654) What conditions determine the opening of a shaft mine with a landing on a trestle, and what are the features in the arrangement of the surface works in this case?

(1655) What conditions determine the opening of a shaft mine with only one landing, and that on natural ground, and what are the features in the arrangement of the surface works in this case?

(1656) In what respects do the requirements at a shaft mine differ from the requirements at other mines in regard to stables, hay, and feed?

(1657) What should be the limits for the foot of the brace between the resultant and underwinding rope?

(1658) What should be the limit of the distance of the center of the engine drums from the center of the shaft, if the center of the head wheels is 60 ft. vertically above the center of the engine drums?

(1659) What are the favorable conditions for the use of a self-dumping cage?

(1660) When can a tipple arrangement be introduced for dumping cars on both sides of the shaft?

(1661) What is the advantage of returning empty cars to the cage by the rear of the shaft, as compared with returning them on the same side from which the loads are removed?

(1662) In what instances is a tipple platform laid with steel plates preferable to track arrangements?

(1663) What are the advantages of car-shifting devices?

(1664) What grades are best suited for loaded and empty tracks at a shaft landing on a trestle?

(1665) What buildings will generally be located on one side of the shaft, what on the other side, and why?

(1666) At a shaft location with landing on the ground, where is the best location for a mechanical lift for returning empty cars to the mine by gravity?

(1667) What branches from the return empty track are needed in the yard, to the various shops, buildings, etc.?

(1668) What will be about the smallest and largest number of cars that a landing at a slope mine will be required to hold?

(1669) What are the grades for outside mine tracks if made gravitating?

(1670) How is the connection with the yard and slope tracks made?

(1671) What is the best arrangement in the surface works for pumping at a slope mine?

(1672) What is the best arrangement for a slope landing for 10 cars, where the distance from the knuckle to the railroad-tracks is 40 feet?

(1673) What should be the least height of a drift mine opening, above the railroad-tracks, so that cars may gravitate to the tipple for dumping?

(1674) What number of cars is required to get out 1,500 tons in 8 hours, where it takes a car 2 hours to make a round trip from the tipple into the mine and return? Capacity of car is 2 tons.

(1675) What is the best arrangement for long tracks outside of a drift, where the ground is limited between the opening and railroad-tracks?

(1676) What are the main features in the surface arrangements at a drift mine?

(1677) What will be the minimum height of a mine opening above the tipple, to be operated by an inclined plane?

(1678) How should mines of less than above height be opened?

(1679) How are tracks arranged at the top and bottom of an inclined plane?

(1680) What is the usual number of cars lowered on an inclined plane and what influences the number?

(1681) How does a steep hillside influence the location of buildings?

(1682) What should be the greatest angle of the plane for lowering cars loaded above the top?

(1683) What should be the greatest angle of the plane for lowering cars loaded level full?

(1684) How is the coal handled on steeper angles?

(1685) What is a safe minimum grade for a gravity plane (*a*) 500 feet long? (*b*) 2,000 feet long?

(1686) How should coal be lowered when it outcrops on a bluff 45 to 65 feet above and near the railroad-tracks? How, if the coal is soft?

(1687) What are the three different styles of tipple dumps?

(1688) What is the best arrangement of tracks for push or cradle tips?

(1689) What is the general arrangement of tracks for a cross-over tip?

(1690) Describe the working of a cross-over tip.

(1691) Where should the center of the car tip be located with respect to the intersection of the line of chute with the tipple platform?

(1692) What should be the angle of the chute for (*a*) lump coal? (*b*) mine-run? (*c*) screening?

(1693) What is the angle of a loading basket (*a*) when shut, and (*b*) when open?

(1694) What are the standard sizes of screen bars?

(1695) What are the standard sizes of their openings?

(1696) Through and over what size bars will nut coal pass?

(1697) How should coal containing very much fine coal be screened?

(1698) What is the advantage of two tipple dumps over one in loading?

(1699) What are the arrangements for loading box cars?

(1700) Where is the weighing of coal generally done?

(1701) How should coal be weighed where it is desired to know the weight of the lump and the screening less than $1\frac{1}{2}$ inches, separately?

(1702) Where is the best location for the weigh-beam?

(1703) How should lump coal that must be broken and picked to remove slate be cleaned?

(1704) How is nut coal containing few impurities cleaned?

(1705) How is nut coal or smaller screening containing much impurity cleaned?

(1706) In what places may rock be dumped from mine-cars?

(1707) If coal breaks easily, how is it dumped?

(1708) How should easily broken coal with much screenings be screened and lowered from small or considerable heights?

(1709) What are the usual curves, grades, and distances from center to center of railroad-tracks?

(1710) What should be the length of each siding where the proportion of the sizes of 1,500 tons daily are as follows:

lump, 70%; nut, 15%; pea, 8%; slack, 7%—cars 34 feet long, holding 30 tons each?

(1711) What is the sharpest curve that should be used for a 14-inch wheel, 20 inches center to center of axle, and 3-foot gauge?

(1712) What should be the grades for loads and empties on long gravitating tracks, and where should a device be introduced for raising the empties for returning them to the mine?

(1713) What grade should be given if the cars start from a standstill?

(1714) If it is desired to maintain tracks at a level, instead of gravitating, how can cars be handled for long distances?

(1715) What will be the length of a mine-car siding for loaded trips where the cars hold 2 tons, trips arriving every 20 minutes, for a day of 8 hours? The output is 1,500 tons per day, and the cars are 8 feet long.

(1716) Under what conditions would iron, wood, and stone tanks be used as reservoirs for the water-supply in preference to an earthwork basin or dam?

(1717) Mention the conditions that determine the relative location of the blacksmith shop, carpenter shop, and machine-shop.

SURFACE ARRANGEMENTS
OF
ANTHRACITE MINES.

EXAMINATION QUESTIONS.

(1718) State the difference, if any, between a drift and a tunnel.

(1719) When the engine is not direct-acting, why is it best to locate the spur-wheel in the center of the hoisting drum ?

(1720) In what way would you make the positive test to determine whether the boiler feed-water contains sulphuric acid ?

(1721) In separating the slate from the coal, what objection is there in having all the slate pickers arranged one above the other in one continuous chute?

(1722) What are the three physical characteristics upon which the operation of separating the slate from the coal by mechanical means is based ?

(1723) In locating an engine to hoist out of a shaft, what are the points to be taken into consideration ?

(1724) In Fig. 1009, Art. **2811**, the difference of elevation between the walls that support the breaker and those that support the tower is 12 feet, and the distance between their centers is 200 feet. The height of the breaker at the point where the dump chute enters the breaker is 90 feet, and the pitch of the dump chute is 3.5 inches to the foot. (*a*) What is the height of the tower to dump chute, and (*b*) what is the length of the dump chute ?

Ans. { (*a*) Height, 136.33 feet.
(*b*) Length of chute, 208.33 feet.

(1725) What is a *telegraph*, and what is it used for ?

(1726) In order to have a well-arranged plant, how should the structures erected upon the surface be arranged, and why?

(1727) State briefly the different methods of handling culm.

(1728) What device to prevent breakage is adopted in connection with rolls?

(1729) What is a proper grade for the empty and loaded tracks leading to and from the breaker, and over which the railroad-cars run?

(1730) What effect have the movable screening bars on the height of the breaker?

(1731) What is the object in using an elliptical gear in operating the piston of a piston jig?

(1732) What are the objections to locating a hoisting-engine in the lower part of the breaker?

(1733) What do you understand by lump-coal screenings?

(1734) What precautions should be taken in locating engines that are used for lowering and hoisting men?

(1735) What does the capacity of a breaker depend upon?

(1736) What does the machinery for conveying the coal in the breaker consist of?

(1737) Name the different kinds of openings which are used at anthracite collieries to furnish the coal to operate the plant.

(1738) Give the pitches of chutes down which the different sizes of wet and dry coal will slide.

(1739) Why is it more expensive to prepare coal by the wet method than by the dry?

(1740) Explain one of the main points to be taken into consideration in choosing the site for a breaker.

(1741) In a slate-picker screen how are the slits or longitudinal openings in slate-picker segments arranged in reference to the screen shaft?

(1742) What are the principal requirements of anthracite winding-engines?

(1743) What are the two methods of removing impurities from coal?

(1744) What are the two main points in connection with jigs in order to have a good separation?

(1745) For what sizes and for what purpose is a pentagonal screen used?

(1746) What constitutes a good steam-brake?

(1747) What are the principal objects aimed at in preparing anthracite coal?

(1748) State the classes into which the machinery for sizing coal is divided.

(1749) State the difference between the two types of coal jigs used in the anthracite region.

(1750) What is the reason for locating breaker engines at some distance from the main structure?

(1751) Why is it that anthracite coal is not marketable as it comes from the mines?

(1752) Give the sizes of the different meshes used in sizing coal.

(1753) In locating a steam plant, what important point must be kept in view?

(1754) What does the capacity of screens depend upon, and what is the effect of increasing the pitch?

(1755) What governs the size of the carpenter and blacksmith shops at an anthracite colliery?

(1756) What arrangement is in use to keep the slits in a cast-iron slate-picker segment from becoming clogged?

(1757) In designing a boiler house, what are some of the main points to be taken into consideration?

(1758) In designing a head-frame, what should the height be, in relation to landing and overwinding points?

(1759) What are the methods in use for driving screens?

(1760) Where is lump coal made?

(1761) What is a *main screen?*

(1762) In what way do the corrugated rolls differ from other rolls?

(1763) What is the best location for the powder house?

(1764) In case of a long line shaft, made up of a number of short pieces of shafting, explain how they are coupled, and show a sketch of the same.

(1765) In what way do the prepared-coal rolls differ from the main rolls?

(1766) Give the name of the screen or screens into which the coal enters after passing through the main screening bars.

(1767) Why is it more economical to use a "gunboat" than an ordinary mine-car on slopes where the pitch is very heavy?

(1768) Make a sketch showing the method usually adopted in locating a fan, when a main slope and a tender slope are used to operate the plant.

(1769) State the classes into which the machinery for preparing anthracite coal is divided.

(1770) State the difference between the two types of movable screens.

(1771) What is the duty of the platform men?

(1772) Where and for what purpose are safety blocks used?

(1773) What type of fan is used in the anthracite region in preference to all others?

(1774) Name in their order the different sizes of coal prepared in the anthracite breaker, commencing with the largest size.

(1775) From the tests thus far obtained from fans, what peripheral speed gives the best results?

(1776) In jigging coal, what is the separation due to?

(1777) From the description and the end view, Fig. 1001, Art. **2783,** make a sketch showing the plan of the machine, so as to show that you understand its operation.

(1778) What is a breaker and what is its purpose?

(1779) What is the difference between the specific gravity of coal and of slate?

(1780) Where are the main rolls or crushers generally located?

(1781) What are corbel blocks used for?

(1782) What is a slush bank?

(1783) Where the power in the breaker is to be transmitted at a slow rate, what kind of belting is preferable?

(1784) Where should the colliery office be located, and why?

(1785) As a rule, how many sets of rolls are there in a breaker?

(1786) Define a mud screen.

(1787) What are loading lips; where and for what purpose are they used?

(1788) In locating a culm pile or heap, where is it advisable not to deposit the culm, and why?

(1789) In comparing the surfaces of a gyrating and a circular screen, what advantage has the former over the latter?

(1790) What do you understand by fixed rolls?

(1791) Describe a slate-picker screen.

A KEY

TO ALL THE

QUESTIONS AND EXAMPLES

CONTAINED IN THE

EXAMINATION QUESTIONS

INCLUDED IN THIS VOLUME.

The Keys that follow have been divided into sections corresponding to the Examination Questions to which they refer, and have been given corresponding section numbers. The answers and solutions have been numbered to correspond with the questions. When the answer to a question involves a repetition of statements given in the Instruction Paper, the reader has been referred to a numbered article, the reading of which will enable him to answer the question himself.

To be of the greatest benefit, the Keys should be used sparingly. They should be used much in the same manner as a pupil would go to a teacher for instruction with regard to answering some example he was unable to solve. If used in this manner, the Keys will be of great help and assistance to the student, and will be a source of encouragement to him in studying the various papers composing the Course.

GASES MET WITH IN MINES.

(355) See Art. **828.**

(356) See Art. **834.**

(357) See Art. **841.**

(358) See Art. **849.**

(359) Carbonic acid gas, because it contains more matter per unit of volume, and is more compact and heavier than marsh-gas. See Art. **830** and Table 19 (Art. **865**).

(360) See Art. **831.**

(361) Applying formula **2,** we have

$$\text{Sp. Gr.} = \frac{175}{62.5} = 2.8. \quad \text{Ans.}$$

(362) See Arts. **835** and **836.**

(363) It drives or tends to drive the molecules apart. See Art. **843.**

(364) See Arts. **846, 847,** and **848.**

(365) See Art. **860.**

(366) See Art. **864.**

(367) See Art. **881.**

(368) The amount of matter in a body, regardless of the space it occupies, is called mass, while the space which the body occupies, regardless of the amount of matter, is called the volume. See Art. **829.**

(369) Applying formula **4,** we have $62.5 \times 1.5 = 93.75$ lb., the weight of 1 cu. ft. of anthracite coal. Hence, the

weight of 1 cu. yd. or 27 cu. ft. = 27 × 93.75 lb. = 2,531.25 lb. Ans.

(370) See Art. **836.**

(371) See Art. **838.**

(372) See Art. **840.**

(373) A compound substance is a substance formed of molecules which are unlike in their nature. See Arts. **836** and **842.**

(374) See Art. **844.**

(375) See Art. **847.**

(376) See Art. **850.**

(377) See Art. **851.**

(378) Applying formula **8,**

$$v_1 = \frac{6 \times 8}{80} = \frac{3}{5} = .6 \text{ cu. ft.} \quad \textbf{Ans.}$$

(379) See Art. **853.**

(380) See Art. **859.**

(381) See Art. **860.**

(382) See Art. **866.**

(383) Applying formula **21,**

$$W = \frac{1.3253 \times 300 \times 30 \times 1.5291}{459 + 70} = 34.48 \text{ lb.} \quad \text{Ans.}$$

(384) See Art. **877.**

(385) See Art. **887.**

(386) See Art. **897.**

(387) See Arts. **902, 903,** and **904.**

(388) See Art. **871.**

(389) See Art. **861.**

(390) See Art. **898.**

(391) See Art. **895.**

(392) See Art. **899.**

(393) Applying formula **12,**

$$20 : W_1 :: 12 : .0763, \text{ or } W_1 = \frac{20 \times .0763}{12} = .1272 \text{ lb. Ans}$$

(394) See Art. **889.**

(395) Applying formula **7,**

$$p_1 = \frac{3 \times 36}{4} = 27 \text{ lb. per square inch. Ans.}$$

(396) See Art. **870.**

(397) See Arts. **882** and **883.**

(398) See Art. **876.**

(399) See Art. **848.**

(400) See Art. **845.**

(401) See Art. **874.**

(402) One pound of bituminous coal, when burned, furnishes 14,400 B. T. U. (see Table 18); hence, 2,000 pounds will furnish $2,000 \times 14,400 = 28,800,000$ B. T. U. Ans.

(403) See Arts. **878** and **879.**

(404) See Art. **873.**

(405) See Art. **856.**

(406) See Art. **839.**

(407) See Art. **841.**

(408) The carbon in the coal $= .88 \times 300$ lb. $= 264$ lb., and since the molecular weight of carbonic acid gas (CO_2) is $12 + 32 = 44$, the carbon in the gas must be $\frac{12}{44}$ of the weight of the gas. Therefore, if 264 lb. of carbon be used to produce carbonic acid gas, 264 lb. will represent $\frac{12}{44}$ of the resulting product. Hence, $\frac{44}{44}$, or the whole of the gas formed, $= \frac{264 \times 44}{12} = 968$ lb. Ans.

See Art. **838** and Table 17.

(409) Dissociation is the disunion of the elements forming a compound. See Art. **837.**

(410) See Art. **834.**

(411) Applying formula **1**, we have

$$\text{Sp. Gr.} = \frac{10}{10 - 8.6} = 7.14. \quad \text{Ans.}$$

(412) To gases only. See Art. **841.**

(413) One molecule of CH_4 yields one molecule of CO_2, and since they are both compound gases, a molecule of each occupies the same volume. Hence, 1,200 cu. ft. of CH_4 will yield 1,200 cu. ft. of CO_2. Ans. See Art. **841.**

(414) Gases, liquids, and solids. See Art. **833.**

(415) See Art. **863.**

(416) See Art. **869.**

(417) See Art. **900.**

(418) See Art. **907.**

(419) See Art. **922.**

(420) See Art. **916.**

(421) Applying formula **20**,

$$V = \frac{15 \times 20 + 9 \times 15}{21} = 20.71 \text{ cu. ft.} \quad \text{Ans.}$$

(422) See Art. **918.**

(423) See Art. **922.**

(424) See Art. **862.**

(425) See Art. **923.**

(426) See Art. **919.**

(427) See Arts. **906** and **915.**

(428) See Art. **890.**

(429) Applying formula **14**, we have

$$p_1 = 14.7\left(\frac{459 + 212}{459 + 50}\right) = 19.38 \text{ lb. per square inch.} \quad \text{Ans.}$$

(430) See Art. **901.**

(431) See Arts. **907** and **910.**

(432) See Art. **913.**

(433) See Art. **915.**

(434) See Art. **891.**

(435) Applying formula **15,** we have

$$P = \frac{.37052 \times 6.5 \times (459 + 84)}{76} = 17.21 \text{ lb. per square inch.}$$

Ans.

(436) See Art. **911.**

(437) See Arts. **912** and **914.**

(438) See Art. **892.**

(439) See Arts. **893** and **894.**

(440) See Art. **900.**

(441) See Art. **896.**

(442) Weight = .0766 × .559 × 100 = 4.28 lb. Ans. See Art. **832.**

(443) Because the square root of the density of carbonic acid gas is greater than that of marsh-gas. See Art. **865.**

(444) The specific gravity of marsh-gas is .559. Using formula **21,**

$$W = \frac{1.3253 \times 650 \times 29.5 \times .559}{459 + 60} = 26.75 \text{ lb.} \quad \text{Ans.}$$

MINE VENTILATION.

(PART 1.)

(445) See Arts. **925**, **927**, and **932**.

(446) (*a*) Using formula **27**,

$$t = \frac{v}{g} = \frac{1{,}876}{32.16} = 58.333 \text{ sec.},$$

the time the ball would require to reach the highest point. Hence, $58.333 \times 2 = 116.66$ seconds, or 1.944 min. Ans.

(*b*) By using formula **28**,

$$h = \frac{v^2}{2g} = \frac{1{,}876^2}{2 \times 32.16} = 54{,}716.66 \text{ ft.},$$

the height the ball will rise. Hence, $54{,}716.66 \times 2 = 109{,}433.3$ ft., the total distance over which the ball will pass. Ans.

(447) See Art. **937**.

(448) See Art. **936**.

(449) Using formula **43**,

$$q = a\,v = 7 \times 7 \times 300 = 14{,}700 \text{ cu. ft. per min.} \quad \text{Ans.}$$

(450) Since the water-gauge is equivalent to a certain pressure, law **(3)**, Art. **980**, may be used. Hence, substituting W and W_1 for p and p_1, respectively,

$$W : W_1 :: q^2 : q_1^2, \text{ or } 2 : 8 :: 15{,}000^2 : p_1;$$

whence, $p_1 = 30{,}000$ cu. ft. Ans.

(451) Law **(15)** evidently applies to this case. Calling the original quantity 1,

$$H : H_1 :: q^3 : q_1^3, \text{ or } 2 : H_1 :: 1^3 : 2^3;$$

whence, $H_1 = 16$ horsepower. Ans.

(**452**) Since $p = 5.2\,W$, $p = 5.2 \times 1.5 = 7.8$ lb. per sq. ft. Applying formula **36**,

$$P = pa = 7.8 \times 6 \times 7 = 327.6 \text{ lb.} \quad \text{Ans.}$$

(**453**) The perimeter of the 6 ft. × 6 ft. airway is $6 \times 4 = 24$ ft.; of the 8 ft. × $4\frac{1}{2}$ ft., $8 \times 2 + 4\frac{1}{2} \times 2 = 25$ ft. Since both airways have the same length, the 6 ft. × 6 ft. airway has less rubbing surface than the 8 ft. × $4\frac{1}{2}$ ft., its perimeter being less. Hence, the 6 ft. × 6 ft. airway will pass the greater quantity of air.

(**454**) Applying the method illustrated in Art. **992**,

$$\sqrt{\frac{a_1^3}{s_1}} = \sqrt{\frac{36^3}{36{,}000}} = 1.1384, \text{ since } a_1 = 6 \times 6 = 36, \text{ and } s_1 = 4 \times 6 \times 1{,}500 = 36{,}000.$$

$$\sqrt{\frac{a_2^3}{s_2}} = \sqrt{\frac{42^3}{46{,}800}} = 1.2582, \text{ since } a_2 = 6 \times 7 = 42, \text{ and } s_2 = (2 \times 6 + 2 \times 7) \times 1{,}800 = 46{,}800.$$

$$\sqrt{\frac{a_3^3}{s_3}} = \sqrt{\frac{30^3}{29{,}700}} = .9535, \text{ since } a_3 = 6 \times 5 = 30, \text{ and } s_3 = (2 \times 6 + 2 \times 5) \times 1{,}350 = 29{,}700.$$

$$\sqrt{\frac{a_4^3}{s_4}} = \sqrt{\frac{25^3}{30{,}000}} = .7217, \text{ since } a_4 = 5 \times 5 = 25, \text{ and } s_4 = 4 \times 5 \times 1{,}500 = 30{,}000.$$

$$sum = 4.0718$$

$$\left.\begin{aligned}
q_1 &= \frac{1.1384}{4.0718} \times 45{,}000 = 12{,}582 \text{ cu. ft. per min. for (1).}\\
q_2 &= \frac{1.2582}{4.0718} \times 45{,}000 = 13{,}905 \text{ cu. ft. per min. for (2).}\\
q_3 &= \frac{.9535}{4.0718} \times 45{,}000 = 10{,}539 \text{ cu. ft. per min. for (3).}\\
q_4 &= \frac{.7217}{4.0718} \times 45{,}000 = 7{,}974 \text{ cu. ft. per min. for (4).}
\end{aligned}\right\} \text{Ans.}$$

$$sum = 45{,}000$$

(**455**) Apply law (**22**). Since $p = 5.2\,W$, $p = 5.2 \times 1 = 5.2$ lb. per square foot. Then, $p : p_1 :: d_1^4 : d^4$, or $5.2 : p_1 :: 5^4 : 6^4$; whence, $p_1 = 12.94$ lb. per sq. ft. Ans.

(**456**) Since quantity and velocity are directly proportional, we may substitute v for q in law (**15**), obtaining $u : u_1 :: v^3 : v_1^3$, or, calling the power originally required 1,

$1 : u_1 :: 4^3 : 8^3$; whence, $u_1 = 8$; i. e., the ratio of increase will be 8 : 1. Ans.

(457) Applying law (**5**), and calling the original pressure and length each 1,

$$p : p_1 :: l : l_1, \text{ or } 1 : p_1 :: 1 : 2;$$

whence, $p_1 = 2$, and the ratio is 2 : 1. Ans.

(458) Applying law (**3**),

$$p : p_1 :: q^2 : q_1^2, \text{ or } 1 : p_1 :: 1^2 : 2^2;$$

whence, $p_1 = 4$, and the ratio is 4 : 1. Ans.

(459) Applying law (**15**),

$$u : u_1 :: q^3 : q_1^3, \text{ or } 1 : u_1 :: 1^3 : 2^3;$$

whence, $u_1 = 8$, and the ratio is 8 : 1. Ans.

(460) Since the volumes are proportional to the absolute temperatures, we may write $v : v_1 :: T : T_1$, T being $459 + 30 = 489$, and T_1 being $459 + 70 = 529$. Hence, $10,000 : v_1 :: 489 : 529$, or $v_1 = 10,818$ cu. ft. per min. Ans.

(461) Since $p = 5.2\ W$, $p = 5.2 \times 2 = 10.4$ lb. per sq. ft. Applying formula **48**,

$$H = \frac{pq}{33,000} = \frac{10.4 \times 120,000}{33,000} = 37.82 \text{ horsepower, nearly.}$$

Ans.

(462) Substituting W and W_1 for p and p_1 in law (**3**),

$$W : W_1 :: q^2 : q_1^2, \text{ or } 3 : W_1 :: 20,000^2 : 30,000^2;$$

whence, $W_1 = 6\tfrac{3}{4}$ in. Ans.

(463) 5 ft. per sec. $= 5 \times 60 = 300$ ft. per min. Applying formula **45**,

$$a = \frac{q}{v} = \frac{8,000}{300} = 26\tfrac{2}{3} \text{ sq. ft.} = \text{area of No. 1 split. Ans.}$$

$$a = \frac{10,000}{300} = 33\tfrac{1}{3} \text{ sq. ft.} = \text{area of No. 2 split. Ans.}$$

$$a = \frac{12,000}{300} = 40 \text{ sq. ft.} = \text{area of No. 3 split. Ans.}$$

$$a = \frac{14,000}{300} = 46\tfrac{2}{3} \text{ sq. ft.} = \text{area of No. 4 split. Ans.}$$

$$a = \frac{16,000}{300} = 53\tfrac{1}{3} \text{ sq. ft.} = \text{area of No. 5 split. Ans.}$$

(464) Applying formula **50**,

$$p = \frac{33{,}000\,H}{q} = \frac{33{,}000 \times 40}{112{,}000} = 11.79 \text{ lb. per sq. ft., nearly.}$$

Hence, $W = \dfrac{11.79}{5.2} = 2.27$ in., nearly. Ans.

(465) Applying the method described in Art. **992**,

$$\sqrt{\frac{a_1^3}{s_1}} = \sqrt{\frac{48^3}{192{,}000}} = .75894$$

$$\sqrt{\frac{a_2^3}{s_2}} = \sqrt{\frac{48^3}{280{,}000}} = .62846$$

$$sum = 1.38740$$

Then, $q_1 = \dfrac{.75894}{1.3874} \times 10{,}000 = 5{,}470$ cu. ft. per min. in A.

$q_2 = \dfrac{.62846}{1.3874} \times 10{,}000 = 4{,}530$ cu. ft. per min. in B.

Ans.

(466) (*a*) The easiest way to work this example is to calculate the ventilating pressure for each split; if all are equal, no regulators will be required, but if some, or all, are different, regulators must be introduced into those splits having the lesser values. The pressure may be calculated by using formula **44** to find the velocity, and then using formula **38**; but an easier way is to use the following formula, which is obtained by transposing terms in formula **q**, Art. **979**:

$$p = \frac{k s q^2}{a^3}.$$

Applying this formula, we have

$p = \dfrac{k s q^2}{a^3} = \dfrac{.0000000217 \times 280{,}000 \times 5{,}000^2}{48^3} = 1.374$ lb. per sq. ft. for A.

$p = \dfrac{k s q^2}{a^3} = \dfrac{.0000000217 \times 150{,}000 \times 10{,}000^2}{50^3} = 2.604$ lb. per sq. ft. for B.

$$p = \frac{ksq^2}{a^3} = \frac{.0000000217 \times 360,000 \times 20,000^2}{72^3} = 8.372 \text{ lb. per}$$
sq. ft. for C.

$$p = \frac{ksq^2}{a^3} = \frac{.0000000217 \times 160,000 \times 15,000^2}{48^3} = 7.064 \text{ lb. per}$$
sq. ft. for D.

Hence, to distribute the air as required by the example, regulators must be placed at A, B, and D. Ans.

(b) After placing the regulators, the pressure will be 8.372 lb. per sq. ft. in all the splits. Therefore, applying formula **48**,

$$H = \frac{pq}{33,000} = \frac{8.372 \times 50,000}{33,000} = 12.685 \text{ horsepower. Ans.}$$

(467) Using formula **21**,

$$W = \frac{1.3253\,VBD}{T},\quad W = \frac{1.3253 \times 30 \times 1 \times 1}{459 + 62} = .076313 \text{ lb.}$$

Now, applying formula **34**,

$$M = \frac{5.2\,G}{W} = \frac{5.2 \times .4}{.076313} = 27.256 \text{ ft. Ans.}$$

(468) In the last example, the weight of a cubic foot of air at 62° F. and 30 inches barometer was found to be .076313 lb.

Hence, applying formula **32**,

$$v = \sqrt{\frac{2gF}{w}} = \sqrt{\frac{2 \times 32.16 \times 2.08}{.076313}} = 41.873 \text{ ft. per sec.} =$$
2,512.38 ft. per min. Ans.

(469) None of the laws will apply to this case, but the example may be worked as follows: Denoting the quantity passed with 15 horsepower by 1, we have for the quantity passed with 36 horsepower [applying law (**15**)],

$$H : H_1 :: q^3 : q_1^3, \text{ or } 15 : 36 :: 1^3 : q_1^3;$$

whence, $q_1^3 = 2.4$, and $q_1 = \sqrt[3]{2.4}$.

Now, applying law (**3**) and substituting W and W_1 for p and p_1, respectively,

$$W : W_1 :: q^2 : q_1^2, \text{ or } .6 : W_1 :: 1^2 : \sqrt[3]{2.4^2};$$

whence, $W_1 = 1.076$ in., nearly. Ans.

(**470**) The rubbing surfaces are $(2 \times 6 + 2 \times 8) \times 8{,}000 = 224{,}000$ sq. ft., and $(2 \times 6 + 2 \times 8) \times 10{,}000 = 280{,}000$ sq. ft. Hence, applying law (**10**), Art. **980**,

$$q : q_1 :: \sqrt{s_1} : \sqrt{s}, \text{ or } 10{,}000 : q_1 :: \sqrt{280{,}000} : \sqrt{224{,}000};$$

whence, $q_1 = 8{,}945$ cu. ft. per min., nearly. Ans.

(**471**) See Arts. **975** and **976.** Since the airways have similar sections,

$$10 : x :: \sqrt[5]{5{,}000} : \sqrt[5]{8{,}000}, \text{ or } x = 10.99 \text{ ft.};$$

also, $10 : 10.99 :: 8 : x$, or $x = 8.792$ ft.

Hence, the required section is 8.792 ft. × 10.99 ft., say 8.8 ft. × 11 ft. Ans.

(**472**) Since the airway is square and $a = 64$ sq. ft., the length of a side $= d = \sqrt{64} = 8$ ft. Representing the pressure by 1, the units of power required would be $u = pq = 1 \times 15{,}000 = 15{,}000$ ft.-lb. Since the power is to remain the same, the pressure for the new airway must be less (the length remaining the same), since the quantity is greater. Hence, $u = p_1 q_1 = p_1 \times 20{,}000 = 15{,}000$, or $p_1 = \frac{15{,}000}{20{,}000} = .75$; i. e., the new pressure is .75 of the original pressure. By using formula **55**, $q = \sqrt{\frac{p\,d^5}{4\,k\,l}}$, whence, $q^2 = \frac{p\,d^5}{4\,k\,l}$; also, $q_1 = \sqrt{\frac{p_1\,d_1^5}{4\,k\,l}}$, whence, $q_1^2 = \frac{p_1\,d_1^5}{4\,k\,l}$. Dividing the first by the second, $\frac{q^2}{q_1^2} = \frac{p\,d^5}{p_1\,d_1^5}$, the denominators canceling out, being equal, or $q^2 : q_1^2 :: p\,d^5 : p_1\,d_1^5$.

Substituting the values of q, q_1, p, d, and p_1,

$$15{,}000^2 : 20{,}000^2 :: 1 \times 8^5 : .75\,d_1^5;$$

whence, $$d_1 = \sqrt[5]{\frac{20{,}000^2 \times 8^5}{.75 \times 15{,}000^2}} = 9.507 \text{ ft.}$$

Hence, the area $= 9.507^2 = 90.38$ sq. ft. Ans.

(473) Applying formula **56**,

$$A = \frac{.0004\,q}{\sqrt{W}} = \frac{.0004 \times 8,000}{\sqrt{\frac{1}{4}}} = 6.4 \text{ sq. ft.} \quad \textbf{Ans.}$$

(474) (*a*) Applying the method described in Art. **992**,

$$\sqrt{\frac{a_1^3}{s_1}} = \sqrt{\frac{30^3}{19,800}} = 1.1678$$

$$\sqrt{\frac{a_2^3}{s_2}} = \sqrt{\frac{36^3}{19,800}} = 1.5350$$

$$\sqrt{\frac{a_3^3}{s_3}} = \sqrt{\frac{24^3}{16,800}} = .9071$$

$$\sqrt{\frac{a_4^3}{s_4}} = \sqrt{\frac{20^3}{12,960}} = .7857$$

$$sum = 4.3956$$

$$\left.\begin{aligned} q_1 &= \frac{1.1678}{4.3956} \times 60,000 = 15,942 \text{ cu. ft. for 1st split.} \\ q_2 &= \frac{1.5350}{4.3956} \times 60,000 = 20,952 \text{ cu. ft. for 2d split.} \\ q_3 &= \frac{.9071}{4.3956} \times 60,000 = 12,381 \text{ cu. ft. for 3d split.} \\ q_4 &= \frac{.7857}{4.3956} \times 60,000 = 10,725 \text{ cu. ft. for 4th split.} \end{aligned}\right\} \text{Ans.}$$

$$sum = 60,000$$

(*b*) Velocity in main split $= 60,000 \div 80 = 750$ ft. per min., since sectional area $= 8 \times 10 = 80$ sq. ft. Applying formula **38** to find the pressure,

$$p = \frac{k s v^2}{a} = \frac{.0000000217 \times 54,000 \times 750^2}{80} =$$

8.24 lb. per sq. ft., nearly.

To find the pressure necessary to force the air through the splits, consider split No. 1.

$$\text{Velocity} = \frac{15,942}{30} = 531.4 \text{ ft. per min.}$$

Applying formula **38**,

$$p_1 = \frac{k\,s_1\,v_1^2}{a_1} = \frac{.0000000217 \times 19{,}800 \times 531.4^2}{30} =$$

4.04 lb. per sq. ft., nearly.

Total pressure $= 8.24 + 4.04 = 12.28$ lb. per sq. ft.

Hence, water-gauge $= \frac{12.28}{5.2} = 2.36$ in., nearly. Ans.

(475) According to law **(15)**,

$$u : u_1 :: q^3 : q_1^3, \text{ or } 1 : u_1 :: 1^3 : 2^3; \text{ whence, } u_1 = 8.$$

That is, the power must be increased to 8 times its original amount in order to double the quantity. Ans.

(476) It is evident, from the conditions of the example, that one airway is 5 times the length of the other. Calling the length of the short airway 1, and the quantity passing through it 1; and the length of the long airway 5, and the quantity passing through it q_1, we have, applying law **(20)**, Art. **980**,

$l : l_1 :: q_1^2 : q^2$, or $1 : 5 :: q_1^2 : 1$; whence, $q_1 = \sqrt{\frac{1}{5}} = .4472$.
Since $q = 1$, $q + q_1 = 1 + .4472 = 1.4472$. Hence,

$q = \frac{1}{1.4472} \times 100{,}000 = 69{,}099$ cu. ft. per min. in short airway.

$q_1 = \frac{.4472}{1.4472} \times 100{,}000 = 30{,}901$ cu. ft. per min. in long airway.

Ans.

(477) (*a*) Applying formula **35**,

$$M = \frac{D(t - t_1)}{459 + t} = \frac{300\,(130 - 50)}{459 + 130} = 40.75 \text{ ft.} \quad \text{Ans.}$$

(*b*) Applying formula **35**,

$$M = \frac{300\,(150 - 50)}{459 + 150} = 49.26 \text{ ft.}$$

The weight of a cubic foot of air at 50° F. and 30 inches barometer is

$$W = \frac{1.3253 \times 30}{459 + 50} = .07811 \text{ lb.}$$

Hence, the pressure per square foot = .07811 × 49.26 = 3.85 lb. Ans.

(478) (*a*) Applying the method illustrated in Art. **992,**

$$\sqrt{\frac{a_1^3}{s_1}} = \sqrt{\frac{36^3}{96,000}} = .69714$$

$$\sqrt{\frac{a_2^3}{s_2}} = \sqrt{\frac{30^3}{66,000}} = .63960$$

$$\sqrt{\frac{a_3^3}{s_3}} = \sqrt{\frac{25^3}{80,000}} = .44194$$

$$sum = 1.77868$$

$q_1 = \frac{.69714}{1.77868} \times 50,000 = 19,597$ cu. ft. per min. for 1st split.

$q_2 = \frac{.63960}{1.77868} \times 50,000 = 17,980$ cu. ft. per min. for 2d split.

$q_3 = \frac{.44194}{1.77868} \times 50,000 = 12,423$ cu. ft. per min. for 3d split.

Ans.

$$sum = 50,000$$

(*b*) Applying formula used in solving example 466,

$p = \frac{ksq^2}{a^3} = \frac{.0000000217 \times 400,000 \times 50,000^2}{100^3} =$ 21.7 lb. per sq. ft.

Therefore, $H = \frac{pq}{33,000} = \frac{21.7 \times 50,000}{33,000} =$

32.88 horsepower for first case. Ans.

$p_1 = \frac{ks_1 q_1^2}{a_1^3} = \frac{.0000000217 \times 96,000 \times 19,597^2}{36^3} =$ 17.15 lb. per sq. ft.

Therefore, $H = \frac{pq}{33,000} = \frac{17.15 \times 50,000}{33,000} =$

25.98 horsepower for second case. Ans.

(479) Applying formula **41,**

$$l = \frac{s}{o} = \frac{54,000}{2 \times 9 + 2 \times 6} = 1,800 \text{ ft.} \quad \text{Ans.}$$

(480) 7 ft. 3 in. = $7\frac{1}{4}$ ft.; 11 ft. 9 in. = $11\frac{3}{4}$ ft. Hence, applying formula **43**,

$q = av = 7\frac{1}{4} \times 11\frac{3}{4} \times 434 = 36,971$ cu. ft. per min., nearly. Ans.

(481) Applying law (**3**),

$p : p_1 :: q^2 : q_1^2$, or $1 : p_1 :: 120,000^2 : 180,000^2$; whence, $p_1 = 2\frac{1}{4}$.

Therefore, the original pressure must be increased $2\frac{1}{4}$ times. Ans.

(482) Applying law (**3**),

$$p : p_1 :: q^2 : q_1^2, \text{ or } 19.2 : p_1 :: 160,000^2 : 120,000^2;$$

whence, $p_1 = 10.8$ lb. per sq. ft. Ans.

(483) For a water-gauge of .9 in., $p = 5.2 \times .9 = 4.68$ lb. per sq. ft. Applying formula **48**,

$$H = \frac{pq}{33,000} = \frac{4.68 \times 70,000}{33,000} = 9.927 \text{ horsepower. Ans.}$$

(484) Applying formula **35**,

$$M = \frac{D(t - t_1)}{459 + t} = \frac{300\,(120 - 45)}{459 + 120} = 38.86 \text{ ft. Ans.}$$

(485) For a water-gauge of .6 in., $p = 5.2 \times .6 = 3.12$ lb. per sq. ft. Applying law (**3**),

$$p : p_1 :: q^2 : q_1^2, \text{ or } 3.12 : p_1 :: 12,000^2 : 24,000^2;$$

whence, $p_1 = 12.48$ lb. per sq. ft. Ans.

(486) Quantity passing in first case is $6 \times 8 \times 300 =$ 14,400 cu. ft. per min. Applying law (**3**),

$$p : p_1 :: q^2 : q_1^2, \text{ or } 4 : p_1 :: 14,400^2 : 24,000^2;$$

whence, $p_1 = 11\frac{1}{9}$ lb. per sq. ft. Ans.

(487) Applying formula **48**,

$$H = \frac{pq}{33,000} = \frac{2.5 \times 20,000}{33,000} = 1\tfrac{17}{33} \text{ horsepower.}$$

Applying law (**15**),

$$H : H_1 :: q^3 : q_1^3, \text{ or } 1\tfrac{17}{33} : H_1 :: 20,000^3 : 25,000^3;$$

whence, $H_1 = 2.959$. Ans.

(488) (*a*) Applying formula **45**,

$$a = \frac{q}{v} = \frac{30,000}{500} = 60 \text{ sq. ft.} \quad \text{Ans.}$$

(*b*) If the current be divided equally, $30,000 \div 2 = 15,000$ cu. ft. per min. must pass in each split, and area of either split $= \frac{15,000}{500} = 30$ sq. ft. Ans.

(*c*) Perimeter of large airway $= \sqrt{60} \times 4 = 31$ ft., nearly. Sum of perimeters of two small airways $= \sqrt{30} \times 4 \times 2 =$ 44 ft., nearly.

Since the lengths of all the airways are equal, it is evident that the two small airways together have more rubbing surface than the large one; hence, they offer more resistance and require greater power in the proportion of 44 : 31, or 1.4194 : 1. Ans.

(489) The easiest method of solving this example is as follows:

By formula **r**, Art. **979**, $s = \frac{u}{k v^3}$ for the first airway, and $s_1 = \frac{u}{k v_1^3}$ for the second airway, since u, the power, is the same for both airways. By transposing the terms v^3 and v_1^3 in their respective equations, $s v^3 = \frac{u}{k}$ and $s_1 v_1^3 = \frac{u}{k}$. Since $s v^3$ and $s_1 v_1^3$ equal the same thing, i. e., $\frac{u}{k}$, they are equal to each other; in other words, $s v^3 = s_1 v_1^3$. For, if $5 \times 6 = 30$ and $2 \times 15 = 30$, it is clearly evident that $5 \times 6 = 2 \times 15$. Since the lengths of the airways are the same, the rubbing surfaces are proportional to the perimeters, and o and o_1 may be substituted for s and s_1. Hence, $o v^3 = o_1 v_1^3$. Now, $o = 2 \times 6 + 2 \times 10 = 32$ ft.; $o_1 = 2 \times 5 + 2 \times 6 = 22$ ft., and $v = \frac{24,000}{6 \times 10} = 400$ ft. per min. Therefore, $o v^3 = o_1 v_1^3$, or $32 \times 400^3 = 22 \times v_1^3$; whence,

$$v_1 = \sqrt[3]{\frac{32 \times 400^3}{22}} = 453.21 \text{ ft. per min.} =$$

velocity in small airway.

Applying formula **43**,

$q_1 = a_1 v_1 = 5 \times 6 \times 453.21 = 13,596$ cu. ft. per min., nearly. Ans.

(490) (*a*) The rubbing surface = $8 \times 3.1416 \times 1,800 =$ 45,239 sq. ft. Applying the formula,

$$p = \frac{k s q^2}{a^3},$$

$$p = \frac{.0000000217 \times 45,239 \times 40,000^2}{(.7854 \times 8^2)^3} =$$

12.36 lb. per sq. ft., nearly. Ans.

(*b*) Applying formula **48**,

$$H = \frac{pq}{33,000} = \frac{12.36 \times 40,000}{33,000} = 14.982 \text{ horsepower. Ans.}$$

(491) The velocity before putting in regulator = $\frac{35,000}{70} = 500$ ft. per min.

The velocity after putting in regulator = $\frac{21,000}{70} = 300$ ft. per min.

Then, as in Art. **998**, $p : p_1 :: v^2 : v_1^2$, or (substituting W and W_1 for p and p_1) $.75 : W_1 :: 500^2 : 300^2$; whence, $W_1 = .27$ in. Therefore, $W - W_1 = .75 - .27 = .48$ in.

Applying formula **56**,

$$A = \frac{.0004 q}{\sqrt{W}} = \frac{.0004 \times 21,000}{\sqrt{.48}} = 12.12 \text{ sq. ft., nearly. Ans.}$$

(492) (*a*) See Art. **941**.
(*b*) See Art. **943**.
(*c*) See Art. **985**.

(493) (*a*) See Art. **997**.
(*b*) See Art. **993**.
(*c*) See Arts. **985** to **987** and Art. **999**.

(494) In Art. **995**, it is shown that $p : p_1 :: s v^2 : s_1 v_1^2$; substituting in this proportion the values given, and replacing p and p_1 by W and W_1,

$.7 : W_1 :: 1 \times 8^2 : 3 \times 10^2$; whence, $W_1 = 3.28$ in. Ans.

(495) (*a*) The rubbing surfaces of the splits are $(2 \times 7 + 2 \times 6) \times 2,000 \times 3 = 156,000$ sq. ft., and $(2 \times 7 + 2 \times 6) \times 5,000 \times 3 = 390,000$ sq. ft. Apply formula **q**, Art. **979**, to the short split.

Since $p = 5.2 \times 2.5 = 13$ lb. per sq. ft., $q = a\sqrt{\frac{p\,a}{k\,s}} = \sqrt{\frac{p\,a^3}{k\,s}} = \sqrt{\frac{13 \times 42^3}{.0000000217 \times 156{,}000}} = 16{,}868$ cu. ft. per min. Ans.

Applying the same formula to the long split,

$$q = \sqrt{\frac{p\,a^3}{k\,s}} = \sqrt{\frac{13 \times 42^3}{.0000000217 \times 390{,}000}} =$$
10,668 cu. ft. per min. Ans.

(*b*) The total quantity $= 16{,}868 + 10{,}668 = 27{,}536$ cu. ft. per min. Applying formula **48**,

$$H = \frac{p\,q}{33{,}000} = \frac{13 \times 27{,}536}{33{,}000} = 10.85 \text{ horsepower.} \quad \text{Ans.}$$

(*c*) As in example 491, $W : W_1 :: v^2 : v_1^2$. But $v = \frac{16{,}868}{42} = 401.6$ ft. per min., nearly; hence, $v_1 = \frac{401.6}{2} = 200.8$ ft. per min. Therefore, $2.5 : W_1 :: 401.6^2 : 200.8^2$; whence, $W_1 = .625$ in. Applying formula **56**, we have, since $16{,}868 \div 2 = 8{,}434$, and $2.5 - .625 = 1.875$,

$$A = \frac{.0004\,q}{\sqrt{W}} = \frac{.0004 \times 8{,}434}{\sqrt{1.875}} = 2.46 \text{ sq. ft., nearly.} \quad \text{Ans.}$$

(496) According to formula **53**, $u = k\,s\,v^2$. Since $v = \frac{q}{a}$, $v^2 = \frac{q^2}{a^2}$ and $u = k\,s\frac{q^2}{a^2}$, or $\frac{s\,q^2}{a^2} = \frac{u}{k}$. Now, as the power is the same for both airways, we have $\frac{s_1 q_1^2}{a_1^2} = \frac{u}{k}$. Hence, $\frac{s\,q^2}{a^2} = \frac{s_1 q_1^2}{a_1^2}$. Assuming that both airways have the same length, we can substitute o and o_1 for s and s_1. Therefore, $\frac{o\,q^2}{a^2} = \frac{o_1 q_1^2}{a_1^2}$. Substituting the values given,

$$\frac{3.1416 \times 18 \times 50{,}000^2}{(.7854 \times 18^2)^2} = \frac{3.1416 \times 6 \times q_1^2}{(.7854 \times 6^2)^2};$$

whence, $q_1 = 8{,}012.5$ cu. ft. per min. Ans.

(497) (*a*) See Art. **973.**
(*b*) See Art. **958.**
(*c*) See Art. **968.**

(498) (*a*) The solution is similar to that given in Art **1000,** $2^5 = 32$; hence, the first figure is 2.

(3) $35 - 32 = 3$. Annexing five ciphers gives 300,000.

(4) $2^4 \times 5$ with four ciphers annexed $= 800,000$; 2^3 with four ciphers annexed $= 80,000$; the sum $= 880,000$.

(5) Since 300,000 will not contain 880,000, the second figure of the root is 0, and the first two figures are 20.

(6) $20^5 = 3,200,000$; $3,500,000 - 3,200,000 = 300,000$.

(7) $20^4 \times 5 = 800,000$; $300,000 \div 800,000 = .37$, or .4.

(8) 20^3 with a cipher annexed $= 80,000$; $80,000 \times .4 = 32,000$, and $800,000 + 32,000 = 832,000$.

(9) $300,000 \div 832,000 = .36$, the fourth and fifth figures. Hence, $\sqrt[5]{35} = 2.036$. Ans.

(*b*) (1) Pointing off into periods gives 642′68937.

(2) $3^5 = 243$; $4^5 = 1,024$; hence, the first figure of the root is 3.

(3) $642 - 243 = 399$; annexing the second period gives 39,968,937.

(4) $3^4 \times 5$ with four ciphers annexed $= 4,050,000$; 3^3 with four ciphers annexed $= 270,000$; the sum $= 4,050,000 + 270,000 = 4,320,000$.

(5) $39,968,937 \div 4,320,000 = 9+$. But 9 is evidently much too large; hence, try 6, making the first two figures of the root 36.

(6) $36^5 = 60,466,176$; $64,268,937 - 60,466,176 = 3,802,761$.

(7) $36^4 \times 5 = 8,398,080$; $3,802,761 \div 8,398,080 = .452$, or .4.

(8) 36^3 with a cipher annexed $= 466,560$; $466,560 \times .4 = 186,624$, and $8,398,080 + 186,624 = 8,584,704$.

(9) $3,802,761 \div 8,584,704 = .443$, say .44.

Hence, $\sqrt[5]{64,268,937} = 36.44$. **Ans.**

MINE VENTILATION.

(PART 2.)

(499) See Art. **1002.**

(500) See Art. **1004.**

(501) See Art. **1005.**

(502) See Art. **1006.**

(503) Using formula **57,**

$$W = \frac{1.3253 \times 30.25}{459 + 350} = .0495.$$

$$.0495 \times 500 \times 3 = 74.25 \text{ lb.} \quad \textbf{Ans.}$$

(504) Using formula **57,**

$$W = \frac{1.3253 \times 29.3}{459 + 32} = .0791 \text{ lb., nearly.} \quad \textbf{Ans.}$$

(505) Using formula **43,** $q = a\,v = 8 \times 10 \times 800 = 64{,}000$ cu. ft. of air at 32° F. Then, using formula **58,** and substituting for T, t, and v their numerical values as given, we have

$$459 + 60 = \frac{V}{64{,}000} \times (459 + 32),$$

and $$519 = \frac{491}{64{,}000}\,V = .007672\,V,$$

and $$V = \frac{519}{.007672} = 67{,}650 \text{ cu. ft., nearly.} \quad \textbf{Ans.}$$

(506) (*a*) Using formula **57,**

$$W = \frac{1.3253 \times 29.8}{459 + 0} = .086043 \text{ lb.,}$$

average weight of 1 cu. ft. downcast air;

$$W = \frac{1.3253 \times 29.8}{459 + 300} = .052034 \text{ lb.,}$$

average weight of 1 cu. ft. upcast air.

$.086043 \times 540 = 46.46322$ lb. per sq. ft., pressure in downcast column.
$.052034 \times 540 = 28.09836$ lb. per sq. ft., pressure in upcast column.

18.36486 lb. per sq. ft., difference. Ans.

(*b*) Using formula **60**, in which we substitute for .077, the average weight of 1 cu. ft. of the downcast air, as obtained in (*a*), we have

$$p = \frac{300 - 0}{459 + 300} \times .086043 \times 540 = 18.365 \text{ lb.} \quad \text{Ans.}$$

(507) Using formula **59**,

$$M = \frac{360 - 60}{459 + 360} \times 200 \times 3 = 219.78 \text{ ft.} \quad \text{Ans.}$$

(508) See Art. **1009.** The isolation of the ribs by side drifts. The isolation of the roof by an air-space between the two arches. The grate area must be proportioned to its work, or vary inversely as the square root of the depth of the shaft. The sectional area over and around the furnace must be proportional to the quantity of air required.

(509) Using formula **61**,

$$s = \frac{34}{\sqrt{250}} = 2.15 \text{ sq. ft. per horsepower.}$$

Using formula **48**, the horsepower is

$$H = \frac{50{,}000 \times 2 \times 5.2}{33{,}000} = 15.76 \text{ H. P., nearly.}$$

$$2.15 \times 15.76 = 33.884 \text{ sq. ft.} \quad \text{Ans.}$$

(510) (*a*) See Art. **1011.** Its object is to isolate the return air of a gaseous mine from the flaming gases and sparks of the furnace.

(*b*) 150 feet.

(511) See Arts. **1012** and **1013.**

(512) (*a*) Any mechanical device for producing an air-current.

(*b*) The centrifugal fan and the steam-jet are familiar examples.

(**513**) The most prominent types of centrifugal ventilators now in use are represented by the Waddle, Schiele, Guibal, and Capell fans. See Art. **1044.**

(**514**) (*a*) By blowing and by exhausting.

(*b*) The efficiencies of each method are practically the same. See Art. **1043.**

(**515**) Blades curve backwards from the direction of their motion, and are so tapered that the breadths of the blades at different distances from the center vary inversely as their distances from the center. See Art. **1045.**

(**516**) The Schiele fan consists of a central disk provided with duplicate sets of blades upon its two sides. Air enters upon each side. The fan is surrounded by a spiral casing, which conducts the air to an evase chimney. See Art. **1046.**

(**517**) It provides a uniformly increasing sectional area about the fan, which gives a uniform velocity to the air-current all around the circumference. See Art. **1046.**

(**518**) To reduce the velocity of discharge and loss of energy. See Art. **1042.**

(**519**) See Art. **1047.**

(**520**) See Art. **1048.**

(**521**) (1) It is safer and (2) it has a uniform efficiency in deep and shallow mines alike.

(**522**) The *furnace* rarefies the air of one shaft by heat, thus causing a difference in pressure, which is the ventilating pressure; the *fan*, by exhaustion or compression, creates the difference in pressure between the intake and discharge openings of a mine. See Arts. **1021** and **1022.**

(**523**) Using formula **64**,

$$v = 18\sqrt{8.41} = 52.2 \text{ ft. per sec.} \quad \text{Ans.}$$

(524) The velocity of the air entering the fan should not exceed 18 feet per second. See Art. **1030.**

(525) See Art. **1030.**

$$\frac{175,000}{2} = 87,500 \text{ cu. ft. on each side.}$$

Using formula **66,**

$$d = .0343\sqrt{87,500} = 10.146 \text{ ft.} \quad \text{Ans.}$$

(526) (*a*) It is the surface of the imaginary cylinder whose diameter is the diameter of the port of entry of the fan, and whose length is the breadth of the fan-blades. See Art. **1031.**

(*b*) The diameter of the port of entry and the width of the blades.

(527) See Art. **1030.**

$$\frac{250,000}{2} = 125,000 \text{ cu. ft. on each side.}$$

Using formula **66,** $d = .0343\sqrt{125,000} = 12.127$ ft. Now, using formula **67,** second case, $b = \frac{1}{2}d = \frac{1}{2} \times 12.127 = 6.06$ ft. Ans.

(528) See Art. **1031.**

$$\sqrt{\frac{153.9384}{.7854}} = 14.0 \text{ ft., diameter of port of entry.}$$

Using formula **67,**

$$b = \frac{1}{4}d = \frac{1}{4} \times 14 = 3.5 \text{ ft.} \quad \text{Ans.}$$

(529) See Art. **1032.**

(530) See Art. **1050.**

(531) See Art. **1051.**

(532) That the velocity is not too low. See Art. **1052.**

(533) (*a*) Too high a velocity of the air-current renders a safety-lamp unsafe, as the flame may be blown through the gauze of the lamp.

(*b*) 450 feet per minute. See Art. **1053.**

(534) (*a*) Doors, stoppings, brattices, curtains, regulators, and overcasts, or bridges.

(*b*) See Art. **1054.**

(535) The water-gauge or manometer, and the anemometer. The water-gauge is used to measure the difference of pressure between the intake and return airways. The manometer is used for the same purpose. The anemometer is used to determine the velocity of the current. See Art. **1057.**

(536) See Arts. **1058, 1059,** and **1060.**

(537) See Arts. **1061, 1062,** and **1063.**

(538) Density of air depends mainly upon two factors, barometric pressure measured by the barometer, and temperature measured by the thermometer. See Art. **1064.**

(539) (*a*) Freezing-point of water $= 0°$ C. and $32°$ F.; boiling-point of water $= 100°$ C. and $212°$ F.

(*b*) $212° - 32° = 180°$ F. $= 100°$ C. See Arts. **1066** and **1067.**

(540) Using formula **76,**

(*a*) $F = \frac{9}{5} \times 350 + 32 = 630 + 32 = 662°$ F. Ans.

(*b*) $F = \frac{9}{5}(-10) + 32 = -18 + 32 = 14°$ F. Ans.

(*c*) $F = \frac{9}{5}(-25) + 32 = -45 + 32 = -13°$ F. Ans.

(541) Using formula **77,**

(*a*) $C = \frac{5}{9}(365 - 32) = \frac{5}{9}(333) = 185°$ C. Ans.

(*b*) $C = \frac{5}{9}(5 - 32) = \frac{5}{9}(-27) = -15°$ C. Ans.

(*c*) $C = \frac{5}{9}(-49 - 32) = \frac{5}{9}(-81) = -45°$ C. Ans.

(542) See Art. **1068.**

(543) See Art. **1069.**

(544) Dip workings, because, as a rule, the *intake* air is cooler than the return air from the workings, and it is natural for the heavier, cool air to flow to the dip, and the lighter air to the rise. See Art. **1071.**

(545) (*a*) Positive air columns are those whose weight acts in the direction in which the current moves.

(*b*) Negative air columns are those whose weight is opposed to the direction of the current. See Art. **1070.**

(546) By ascensional ventilation we understand such a method of ventilation that the general course of the current will be towards the rise.

(547) The algebraic sum of the weights of the positive and negative air columns in any mine is always equal to the weight of the *motive* column, or the pressure per square foot producing the flow of air. See Art. **1072.**

(548) The main feature in such a case is the manner of splitting the air at each pair of cross-entries, when these entries are sufficiently developed to warrant the expense. See Art. **1074.**

(549) The velocity of the divided current, which must not fall below 3 or 4 feet per second in non-gaseous mines and 5 or 6 feet per second where gas is given off. See Art. **1074.**

(550) In non-gaseous mines the haulage roads should be made the return airways for two reasons. See Art. **1075.** In gaseous mines, haulage is done upon the intake airway in order to lessen, as far as possible, liability to explosion. See Art. **1077.**

(551) That the ventilation shall be ascensional.

(552) (1) The establishment of the air-current. (2) The direction of the current should not be altered, except upon most urgent demand. (3) Repairs of stoppings, doors, and brattices must be made rapidly, as no great advance can be made ahead of the air. See Art. **1081.**

(553) See Art. **1083.**

(554) Care must be taken to begin sealing off a fire at its side next to the return air, and work towards the intake, as then there is less opportunity for the entrapping of pure air, which would give rise to an explosion under certain conditions.

HOISTING AND HOISTING APPLIANCES.

(1585) See Art. **2452.**

(1586) Electric motors and steam or compressed-air engines.

(1587) See Art. **2463.**

(1588) See Arts. **2454** and **2455.**

(1589) See Arts. **2458** and **2459.**

(1590) See Art. **2460.**

(1591) See Art. **2462.**

(1592) See Art. **2464.**

(1593) See Art. **2464.**

(1594) See Arts. **2465** to **2469.**

(1595) See Art. **2465.**

(1596) See Art. **2467.**

(1597) See Arts. **2466** and **2467.**

(1598) See Art. **2468.**

(1599) See Art. **2469.**

(1600) See Art. **2470.**

(1601) See Art. **2470.**

(1602) See Art. **2471.**

(1603) The minimum diameter of drum is 60 times the diameter of the rope.

(1604) (*a*), (*b*), and (*c*) See Art. **2471.**
(*d*) See Art. **2473.**

(1605) (*a*) Assume the weight of the rope to be 2,000 lb. Then, the load on the rope is

Material	3,000 lb.
Car	1,800 lb.
Cage	2,400 lb.
Rope	2,000 lb.
Total	9,200 lb.

Using a factor of safety of 10, the breaking load is 92,000 lb. = 46 tons. Referring to Table 46, a 1-inch plow-steel rope with 19 wires to the strand has a breaking load of 47 tons; its weight is 1.58 lb. per ft. In this case, the weight is $1,200 \times 1.58 = 1,896$ lb., which is quite close to the assumed weight. Therefore, a 1-inch rope should be used. Ans.

(*b*) The smallest allowable drum has a diameter 60 times that of the rope, or 60 in. = 5 ft. Ans.

(1606) Using two cages, the gross load is

Material	3,000 lb.
2 cars	3,600 lb.
2 cages	4,800 lb.
Rope	1,896 lb.
Total	13,296 lb.

The net load is

Material	3,000 lb.
Rope	1,896 lb.
Total	4,896 lb.

Actual load = net load + 10% of gross load = $4,896 + .10 \times 13,296 = 6,225.6$ lb. Ans.

(1607) See Art. **2474.**

(1608) The working diameter of the drum is $60 + 1 = 61$ in. $= \frac{61}{12}$ ft.

$$6,225.6 \times \frac{61}{12} \times 3.1416 = 99,421.6 \text{ ft.-lb.} \quad \text{Ans.}$$

(1609) Using formula **211**,

$$D = 1.97\sqrt[3]{\frac{99{,}421.6}{48.76 \times 1.5}} = 21.82, \text{ say } 22 \text{ in.} \quad \text{Ans.}$$

Stroke $= 22 \times 1.5 = 33$ in. Ans.

(1610) Area of piston $= 12^2 \times .7854 = 113.1$ sq. in. The piston travels per revolution $\frac{24}{12} \times 2 = 4$ ft. Total pressure on piston $= 113.1 \times 40$. Work = total pressure $\times$ distance traveled by piston $= 113.1 \times 40 \times 4 = 18{,}096$ ft.-lb. Ans

(1611) See Arts. **2471** and **2472.**

(1612) (*a*) Using formula **211**,

$$D = 1.97\sqrt[3]{\frac{36{,}000}{40 \times 2.5}} = 14 \text{ in.} \quad \text{Ans.}$$

Stroke $= 14 \times 2.5 = 35$ in. Ans.

(*b*) Area of piston $= 14^2 \times .7854 = 153.938$ sq. in.
Length of crank $= \frac{35}{2} = 17\frac{1}{2}$ in.
Turning moment = total pressure on piston $\times$ length of crank $= 153.938 \times 40 \times 17\frac{1}{2} = 107{,}757$ in.-lb. Ans.

(1613) See Art. **2476.**

(1614) It may be smaller. See Art. **2477.**

(1615) Larger. See Art. **2478.**

(1616) See Art. **2480.**

(1617) See Art. **2482.**

(1618) See Art. **2484.**

(1619) Least diameter of drum $= 1\frac{1}{2} \times 60 = 90$ in.
Effective diameter $= 90 + 1\frac{1}{2} = 91\frac{1}{2}$ in. $= 7\frac{5}{8}$ ft.
Circumference $= 7\frac{5}{8} \times 3.1416 = 24$ ft., nearly.

$$\text{Number of turns} = \frac{1{,}800}{24} = 75.$$

Adding 5 turns for friction and for possible overwinding, the number of turns is 80.
Width for each turn $= 1\frac{1}{2} + \frac{1}{4} = 1\frac{3}{4}$ in.

$$80 \times 1\tfrac{3}{4} = 140 \text{ in.} = 11 \text{ ft. } 8 \text{ in.} \quad \text{Ans.}$$

(1620) See Art. **2488.**

(1621) See Arts. **2488** and **2491.**

(1622) See Arts. **2485** and **2526.**

(1623) Assume, first, that the rope weighs 2,000 lb. Then, the load on the rope is

Material	4,000 lb.
Car	3,000 lb.
Cage	3,200 lb.
Rope	2,000 lb.
Total	12,200 lb.

Using a factor of safety of 10, the breaking load is 61 tons, which, from Table 46, requires a $1\frac{3}{8}$-inch rope, the weight of which is 3 lb. per foot. The weight of rope is $3 \times 800 = 2{,}400$ lb., which adds 400 lb. to the previous total weight, making it 12,600 lb., or 6.3 tons. The breaking load is, therefore, 63 tons, and the $1\frac{3}{8}$-inch rope is correct. The minimum diameter is $1\frac{3}{8} \times 60 = 82\frac{1}{2}$ in. $= 7$ ft., nearly. Ans.

Using formula **213,**

$$D = \frac{7\,(4{,}000 + 2 \times 6{,}200 + 2 \times 2{,}400)}{4{,}000 + 2 \times 6{,}200} = 9.05 \text{ ft., say } 9 \text{ ft.}$$ Ans.

(1624) See Art. **2492.**

(1625) See Arts. **2498** to **2503.**

(1626) See Art. **2505.**

(1627) See Arts. **2507** and **2510.**

(1628) See Art. **2511.**

(1629) See Art. **2512.**

(1630) See Art. **2516.**

(1631) See Art. **2514.**

(1632) See Art. **2516.**

(1633) See Art. **2516.**

(1634) See Arts. **2518** and **2520.**

(1635) See Art. **2497.**

(1636) See Art. **2529.**

(1637) See Arts. **2530** and **2531.**

(1638) See Art. **2532.**

(1639) See Art. **2535.**

(1640) See Art. **2535.**

(1641) See Art. **2537.**

(1642) See Arts. **2538** and **2539.**

(1643) See Art. **2540.**

(1644) See Art. **2543.**

(1645) See Art. **2546.**

(1646) See Art. **2547.**

(1647) See Art. **2549.**

(1648) See Art. **2549.**

(1649) See Art. **2559.**

(1650) See Art. **2560.**

(1651) See Arts. **2563** and **2564.**

SURFACE ARRANGEMENTS
OF
BITUMINOUS MINES.

(1652) By limiting the size of the opening through which the cars are hauled Small seams necessitate the use of low cars, and a bad roof necessitates narrow headings, and, consequently, comparatively narrow cars. See Art. **2568.**

(1653) See Art. **2569** and Figs. 947 and 948.

(1654) See Arts. **2572** to **2575** and Fig. 947.

(1655) See Arts. **2576** to **2579** and Fig. 948.

(1656) See Art. **2611.**

(1657) See Art. **2583.**

(1658) From 1 to 1.3 times the vertical height of the center of the sheaves above the center of the drum, or from 60 to 78 feet. See Art. **2586.**

(1659) See Art. **2589.**

(1660) See Art. **2589.**

(1661) See Art. **2596.**

(1662) See Art. **2597.**

(1663) See Art. **2600.**

(1664) See Art. **2601.**

(1665) See Figs. 947 and 948 and Art. **2602.**

(1666) See Art. **2606.**

(1667) See Art. **2607.**

(1668) See Art. **2610.**

(1669) See Arts. **2613** and **2620.**

(1670) See Art. **2614.**

(1671) See Art. **2616.**

(1672) See Arts. **2618** and **2619.**

(1673) See Art. **2622.**

(1674) $\frac{8}{2} = 4$ trips for each car per day. $4 \times 2 = 8$ tons for each car per day. Hence, $\frac{1,500}{8} = 187.5$, or 188 cars. Ans.

(1675) See Figs. 963 and 964 and accompanying description.

(1676) See Art. **2625.**

(1677) See Art. **2630.**

(1678) See Arts. **2630** and **2640.**

(1679) See Figs. 965 and 966 and accompanying descriptions.

(1680) See Art. **2631.**

(1681) See Art. **2631.**

(1682) See Art. **2634.**

(1683) See Art. **2634.**

(1684) See Art. **2634.**

(1685) See Art. **2638.**

(1686) See Arts. **2639** and **2640.**

(1687) See Art. **2644.**

(1688) See Art. **2647.**

(1689) See Art. **2649.**

(1690) See Art. **2650.**

(1691) See Art. **2654.**

(1692) See Art. **2654.**

(1693) See Art. **2660.**

(1694) See Art. **2657.**

(1695) See Art. **2655.**

(1696) See Art. **2655.**

(1697) See Art. **2669.**

(1698) See Art. **2670.**

(1699) See Art. **2663.**

(1700) See Art. **2664.**

(1701) See Art. **2664.**

(1702) See Art. **2664.**

(1703) See Art. **2665.**

(1704) See Art. **2665.**

(1705) See Art. **2665.**

(1706) See Art. **2666.**

(1707) See Art. **2667.**

(1708) See Art. **2669.**

(1709) See Art. **2672.**

(1710)

	Tons.
Output of lump coal = $1{,}500 \times .70 =$	1,050
Output of nut coal = $1{,}500 \times .15 =$	225
Output of pea coal = $1{,}500 \times .08 =$	120
Output of slack coal = $1{,}500 \times .07 =$	105

The lengths of sidings are:

$$\left.\begin{aligned} &\text{Lump, } \frac{1{,}050 \times 34}{30} = 1{,}190 \text{ ft.} \\ &\text{Nut, } \frac{225 \times 34}{30} = 255 \text{ ft.} \\ &\text{Pea, } \frac{120 \times 34}{30} = 136 \text{ ft.} \\ &\text{Slack, } \frac{105 \times 34}{30} = 119 \text{ ft.} \end{aligned}\right\} \text{Ans.}$$

See Art. **2673.**

(1711) See Art. **2674.**

(1712) See Arts. **2675** and **2676.**

(1713) See Art. **2675.**

(1714) See Art. **2676.**

(1715) 8 hr. $= 8 \times 60 = 480$ min.
$480 \div 20 = 24$ trips.
$1{,}500 \div 24 = 62$, the number of tons hauled per trip.
$62 \div 2 = 31$, number of cars per smallest trip.
$31 \times 8 = 248$ ft., smallest length of siding.
248 ft. $\times 2 = 496$ ft., proper length. Ans. See Art. **2677.**

(1716) See Art. **2684.**

(1717) See Arts. **2695** to **2698.**

SURFACE ARRANGEMENTS

OF

ANTHRACITE MINES.

(1718) A drift is driven in the coal seam, while a tunnel is driven across the measures. See Arts. **2720** and **2721.**

(1719) See Art. **2747.**

(1720) See Art. **2797.**

(1721) See Art. **2862.**

(1722) See Art. **2802.**

(1723) See Arts. **2738** and **2739.**

(1724) (*a*) Since the inclination of the dump chute is $3\frac{1}{2}$ inches per foot, or $\frac{7}{24}$, the tower end is $\frac{7}{24} \times 200 = 58\frac{1}{3}$ feet higher than the breaker end. Therefore, the tower end of the chute is $90 + 58\frac{1}{3} = 148\frac{1}{3}$ feet above the wall or the breaker, or $148\frac{1}{3} - 12 = 136.33$ feet above the wall of tower. Ans.

(*b*) The length of the chute is the length of the hypotenuse of a right-angled triangle of which the base is 200 feet and the altitude is the rise, $58\frac{1}{3}$ feet. The length is, therefore, $\sqrt{200^2 + 58\frac{1}{3}^2} = 208.33$ ft. Ans.

(1725) See Arts. **2731** and **2904.**

(1726) See Art. **2724.**

(1727) See Arts. **2778** to **2792.**

(1728) See Art. **2858.**

(1729) See Arts. **2793** and **2794.**

(1730) See Art. **2826.**

(1731) See Art. **2868.**

(1732) See Art. **2740.**

(1733) See Art. **2893.**

(1734) See Art. **2743.**

(1735) See Art. **2806.**

(1736) See Art. **2875.**

(1737) See Art. **2731.**

(1738) See Art. **2877.**

(1739) See Art. **2803.**

(1740) See Art. **2729.**

(1741) At right angles. See Art. **2895.**

(1742) See Art. **2745.**

(1743) See Art. **2803.**

(1744) See Art. **2867.**

(1745) See Art. **2840.**

(1746) See Art. **2749.**

(1747) See Art. **2800.**

(1748) See Art. **2823.**

(1749) See Arts. **2866** to **2874.**

(1750) See Art. **2750.**

(1751) See Art. **2799.**

(1752) See Art. **2835.**

(1753) See Art. **2731.**

(1754) See Art. **2842.**

(1755) See Art. **2770.**

(1756) See Art. **2896.**

(1757) See Art. **2735.**

(1758) See Art. **2758.**

(1759) See Art. **2833.**

(1760) See Arts. **2829** and **2924.**

(1761) See Art. **2830.**

(1762) See Art. **2856.**

(1763) See Art. **2771.**

(1764) See Art. **2889.**

(1765) See Art. **2854.**

(1766) See Art. **2925.**

(1767) See Art. **2817.**

(1768) See Art. **2764.**

(1769) See Art. **2822.**

(1770) See Art. **2828.**

(1771) See Art. **2924.**

(1772) See Art. **2763.**

(1773) The Guibal. See Art. **2766.**

(1774) See Art. **2829.**

(1775) See Art. **2767.**

(1776) See Art. **2866.**

(1777) The description is given in Art. **2783**

(1778) See Art. **2723.**

(1779) See Art. **2866.**

(1780) See Art. **2851.**

(1781) See Art. **2762.**

(1782) See Art. **2782.**

(1783) See Art. **2883.**

(1784) See Art. **2773.**

(**1785**) See Art. **2849.**

(**1786**) See Art. **2830.**

(**1787**) See Art. **2884.**

(**1788**) See Arts. **2778** and **2792.**

(**1789**) See Art. **2846.**

(**1790**) See Art. **2857.**

(**1791**) See Arts. **2831** and **2896.**